Electrochemical and Metallurgical Industry

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CONTENTS.

The Utilization of Blast Furnace Gases for Power Purposes.

That the blast furnace, besides being primarily an iron producer, is at the same time an efficient gas generator, was early recognized. Thus J. B. Budd wrote in 1848 the prophetical words: "It would appear to be more profitable to employ a blast furnace, if as a gas generator only, even if you smelted nothing in it and carried off its heated vapors by flues to your boilers and stoves, than to employ a separate fire to each boiler and each stove." Metallurgical practice has followed to a certain extent the lines indicated in this prophecy. The blast furnace gases which were originally wasted and are for this reason often called "waste gases" even to-day, have long been usefully employed for heating hot-blast stoves, for raising steam in boilers, for calcining ore and for general heating purposes, like drying molds, etc. Further, before burning the gases, they may undergo, in certain cases, a treatment in condensing apparatus, so as to win tar and ammonia as valuable by-products. However, in recent years, with the growth of the gas engine, the necessity arose to get cheap gas, and while excellent progress has been made in the design of gas producers, yet the question naturally presented itself to use the blast furnace gases in gas engines.

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Europe has taken the lead in this development, and there are now some 300,000 hp. produced abroad in gas engines supplied with blast furnace gases. This country has been slow to follow, for the commercial reason that fuel is cheaper and labor more expensive here than in Europe. Nevertheless, the economy of the system was so manifest from the results obtained in European plants that this possible saving could not be overlooked by our iron masters. The pioneer plant in this country was that of the Lackawanna Steel Co., the gas plant being installed by the De La Vergne Machine Co. It was a courageous undertaking, using a large number of 2,000 and 1,000-hp. engines driving alternators. The first set was started on Jan. 2, 1903. Naturally, some difficulties were first experienced, partly due to certain inherent difficulties of operating large gas-driven alternators, and partly due to troubles experienced in the gas plant. It is now unnecessary to argue whether these early troubles were due to certain points in the design of the gas engines—as was claimed from one side—or to the condition of the gas when delivered to the engines—as was claimed by the other side. This huge pioneer plant is now in successful operation, and this fact has undoubtedly acted as a stimulus toward further progress.

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The United States Steel Corporation has now taken the matter up, and has done it in no half-hearted way. We recently mentioned that gas-electric generating plant to the amount of 15,000 hp. has been ordered for the Illinois Steel

Co., 4,000 hp. for the Homestead plant of the Carnegie Steel Co., 35,000 hp. for the Indiana Steel Co., further, 3,500 hp. gas blowing-engines for the Homestead plant. All these engines will be furnished by the Allis-Chalmers Co., while the Westinghouse Machine Co. will provide eight 3,000-hp. gas blowing-engines for the Gary plant of the Indiana Steel Co. For several months a 350-hp. gas engine of this type has been in regular operation on blast furnace gas at the Edgar Thompson Works in Pittsburg, generating electric power for motor-driven foundry machinery. This engine was installed largely for experimental purposes, and has proven very successful. While this is being written, this engine is completing a thirty-day continuous load and duty test. operating 24 hours per day and seven days per week during this period. As other instances of developments in this direction may be mentioned the 800-hp. gas plant of the American Steel & Wire Co., and the 1,350 hp. gas plant of the Pittsburgh Plate Glass Co. at Crystal City, Mo.

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Now, about the saving which is attainable by the utilization of blast furnace gases for the development of electric power. Of perhaps the three best papers which have been written on this subject in recent years in this country, two were contributed to our Vol. III., supplementing each other in a very interesting manner: Mr. F. du P. Thomson, who was intimately connected with the pioneer blast-furnace gas plant of this country, took the stand of the progressive, yet conservative engineer, who claims only as much as he can safely guarantee in the present state of the art, and who knows not only the possibilities but also the drawbacks; Mr. A. J. Rossi contributed an equally clever and interesting article. but written in the mood of the enthusiastic and optimistic pioneer. A position somewhat between the two, with respect to the possibilities in exact figures, was taken by Mr. Freyn in his able paper, read some time ago before the Western Society of Engineers. Without any discourtesy to others, it is probably safest to base the following discussion on Mr. Thomson's figures. First, the requirements of the furnace plant itself must be met for heating and compressing the blast and for operating the auxiliaries (pumps, hoists, etc.). The remainder represents the surplus available for power, and Mr. Thomson finds that the amount of surplus heat available for power per ton of iron produced per hour in the blast furnace is sufficient to develop 468 hp. (with gas engines which have a thermal efficiency of 25 per cent).

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According to the annual statistical report of the American Iron and Steel Association, issued June 30. 1906, the total production of pig iron in the United States was 16,497,-033 long tons in 1904, and 22,992,380 or, say, 23,000,000 tons in 1905. On the basis of the above figure of Mr. Thomson, the surplus power available with an annual output of 23,000-000 of pig iron will be 1,225,000 hp. It will be interesting to make some comparisons. According to the last census report (1902), the total power in all electric central stations in this country was 1,800,000 hp. Further, to come to a subject of more interest just at present, the total hydraulic power of Niagara Falls, if fully developed, would be 3,500,000 hp. Mr. H. W. Buck, of the Niagara Falls Power Co., in an able recent article in the *Outlook*, illustrated what the maintenance of Niagara Falls as a national spectacle means from the commercial point of view. It is equivalent to a great conflagration in which 50,000,000 tons of coal are annually consumed for spectacular purposes, since this is the amount of coal which would have to be burnt to produce 3,500,000 hp. with steam engines. Without entering into a discussion of the aesthetic versus the economic value of Niagara Falls, we may point out that the waste of the blast furnace gases in 1905 in this country was equivalent to a conflagration of more than one-third the size just mentioned. Since with blast furnace gases no aesthetic reasons are to be considered. it is really difficult to see why they should be wasted and have been wasted for such a long time. The recent rapid progress is, therefore, perfectly natural. Incidentally it must aid the commercial development of electric steel processes. As Dr. J. W. Richards once pointed out, if coal would be taken and used in a blast furnace for reducing iron ore, and if all the furnace gases were utilized in gas engines to run dynamos, and an electric refining furnace was used for making crucible steel, less fuel would be consumed altogether than is actually used for making steel by the crucible process at the present time.

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The Flow of the Lines of Current in Electro-chemical Systems.

Storage batteries being a systematic means for changing electrical into chemical energy and vice versa, differ in their behavior in no way from other electrolytic cells and, therefore, form distinctly part of electrochemistry, considered as a science. Yet the practical engineer who looks at industrial applications and not at science for classification, will undoubtedly put storage batteries together with electrical apparatus and not with electrochemistry. It would be a pity if for this reason Mr. Schoop's article, concluded in our present issue, should escape the attention of practical electrochemical engineers. As it stands, Mr. Schoop's investigation was made for the special purpose of determining by experiment the configuration of the lines of current in storage batteries. But it is manifest that the same method may be applied to electrolytic cells in general, and that since the distortion of the lines of current can produce particular and important results in storage batteries, analogous results will be found in other electrolytic cells. Nor is this all. The study of the paths which the lines of current take, is just as important for electric furnace work. For it is clear that the whole problem is intimately connected with Ohm's law, and there is no essential difference between the application of Ohm's law to electrolytes and to metallic conductors. Mr. Schoop is right in saying that Ohm's law in its special form, as used for metallic wires (namely, e. m. f. = current multiplied by resistivity and length, divided by cross-section), cannot be applied to electrolytic cells. But the reason is simply the way how we usually arrange an electrolytic cell and its connections with the external circuit. Ohm's law in its most general form is equally valid for metallic conductors and for electrolytes.

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The manifest advantage of determining and plotting dia-

grammatically the configuration of the system of the lines of current is that it enables us to see at a glance the variation of the current density throughout the cell, and it is an experimental fact that by varying the current density we are able to change to a certain extent the electrochemical action which will take place. In an electric furnace the reaction at a certain point depends on the temperature, and the latter depends on the watts consumed per cubic centimeter at that point, and these equal the square of the current density multiplied by the electric resistivity of that point. Since we have discussed in detail in an editorial note in our February issue the rôle which current density plays in electrolytic and in electric furnace processes, no further discussion of it is here necessary. We may, however, say that the application of Mr. Schoop's method of determining the flow of the lines of current would presumably be found specially useful in certain cases of electroplating and in electrolytic refining practice.

Professional Ethics.

Dr. Schuyler Skaats Wheeler, in his recent residential address before the American Institute of Electrical Engineers, discussed professional ethics—a subject most germane to the times. The past decade's great progress in industrialism has disturbed the settled relations of things. New and unlooked-for conditions have been obtruded into prominence. Such phenomena are described in two ways, according to the individual's hepatic efficiency. By the optimistic as "growin' pains," leading to a yet more splendid adolescence. By the pessimistic, as the rumbling of an eruption of a political Vesuvius. General considerations of ethical questions leave the greatest freedom for doubt and always lead to indefinite conclusions. The Latin dictum, "errare est humanum," is especially predicatory of personal and public morality. Where man, a social being, should guide his conduct purely from an egotistical standpoint, and where purely altruistic doctrines should sway him, is a hard question. This is due to his mixed nature just as is the distinction between the paternalistic theory of government as against the "laissez faire" theory. Mistakes are made in seeing the facts. Mistakes are made in reasoning from the observed facts. Mistakes are made from the happening of the "unexpected." The emotions and inherited instincts cause further mistakes. In short, there is the greatest chance for error, because we are not cold-blooded "reasoning machines," but plain human beings.

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In the relation between client and technical advisor, mixed motives, differentiating between the engineer's personal interest and his fiduciary interest, are the chief causes of trouble. In America, we are frankly individualistic in our usual everyday life. In Europe, however, vestiges of the feudal relations are everywhere manifest. They are best exemplified in a new phase of life—the implicit confidence and trust the French inventor gives the French banker. Thus the American engineer has not the clean-cut and rigid system of ethics that the older professions have, because his is a newer calling in a newer land, and being so, is mainly individualistic. But this individualism must be counteracted by adherence to Lord Bacon's maxim: "I hold every man a debtor to his profes-

sion, from the which as men of course do seek contenance and profit, so ought they of duty to endeavor themselves to be a help and ornament thereunto." This inspiring sentence rings as true to-day as it did four hundred years ago.

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In metallurgical engineering the great changes working havoc in the old order of things, the all-pervasive expansion with its consequent improvements and shifting of the engineer's connections, have naturally brought about unstable conditions. Young and ambitious engineers often designedly change positions in order to gain a wider practical experience, as does the German artisan in his "Wanderjahre." Not infrequently the young engineer is placed in charge of experimental work because of some special training and because also of his enthusiasm. Experience thus gained is undoubtedly a form of capital in the intangible shape of a good memory, though quite tangible when it comes to the asking of good fees. This knowledge represents his professional stick-in-trade to a large extent, but should be left with his employer in the form of accurate data.

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The direct relation between capitalist and engineer is the most important in the domain of professional ethics. The man of business has a superior knowledge of affairs and the human traits, all summed up in the words "commercial sagacity." This superior knowledge is sometimes used as was superior physical strength in an earlier era. And consequent to the widespread belief that "business success is often a survival of the slickest," there is a feeling on the part of the technical man that there is an unfair distribution of profits more legitimate than the same feeling exhibited by workmen. This results in a failure to reach the highest pitch of industrial efficiency. The course of Andrew Carnegie of taking up the practical men as protegés and partners was potent for the success of his plants. Well-measured enthusiasm cannot be conquered. Harmony and "team-play" are irresistible, and Mr. Carnegie's fairness and generosity to his engineers were more mighty factors of his victory over the Western steel plants than were his holdings of natural gas lands and coking-coal fields.

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The complexity of this simple subject compounds itself. But all can be reduced to the simple terms of Lord Bacon's maxim of professional indebtedness to the past knowledge which has made the profession possible. This calls upon the engineer to publish such well-digested opinions as professional papers before the proper technical society. Sometimes, of course, this is impossible. Where to draw the line between individual interests, the employer's interests and society's interests is hard, as it is always hard to draw any sharp line. Much is left to the "personal equation" and the ability to apply general principles derived of experience to a given case, on which, in fact, success in law, medicine or any other profession depends. But in these days of rapid and sensitive public opinion, altruistic honor and "enlightened self-interest" seem rarely to differ. This is true no less of professional ethics than of all other questions of private and public morality, and is not inapplicable to life insurance standards.

The Iron and Steel Market.

At the few points where improvement was possible the American iron trade has shown it in the past two months. The entire market is now in as strong and healthy a condition as can be conceived. Erratic as the curve of market prosperity in the American iron trade appears, when but portions of it are viewed, it is a fact that a general law can be deduced to which the observations will conform with remarkable accuracy, and in opening this series of reports it is perhaps not unfitting that a few words should be devoted to this subject.

The law calls for a complete cycle in a period of twenty years, which may be condensed by special conditions. The war of the rebellion reduced the period to that from 1856 to 1873, but from the latter date to 1893 the full twenty years intervened. Market history since 1893 has most closely duplicated that which followed 1873. There is a period of stagnation of five or six years, prices finding their lowest level towards the close, as costs of production have been successively pared at this point and that. The curve of prices leaps suddenly upwards to a maximum for the whole period (1879 and 1899), and as suddenly falls almost to the minimum; it then rises much less steeply and to a somewhat less height, falling at about the same angle, to again turn upward at a smaller angle and to a less height, the waves thereafter being of less and less amplitude, until a great drop winds up the cycle. The fact which most impresses the one who observes it, and is the cause of the greatest confusion to the one who fails to discern it, is that the amplitude continually decreases while the wave length increases. The ordinary business man looks for a movement as great and as sudden as the immediately preceding one, and is disappointed.

In the second quarter of this year various slight disturbances occurred in the market, and each was taken in some quarters to presage a general decline, but they have all disappeared, and the present market is, as stated, thoroughly strong and healthy. It is not the time for either a great depression or a great advance.

PIG IRON.

The pig iron market during the second quarter of the year was characterized by weak spots, which developed here and there as to both time and place. During July the movement has been uniformly towards greater strength. In the South the market had broken to $13.00, f. o. b. Birmingham, for No. 2 foundry, and has reacted to $13.50, through large sales and a reduction in output caused by physical necessities. In the North, Bessemer, through the inability of merchant furnaces to buy additional Bessemer ores for this season, and basic, through the limited number of furnaces equipped with chill casting apparatus, have become very scarce, while foundry pig has strengthened in all Northern districts. The turn in pig iron was doubtless caused by a slight reduction in output, through the necessary blowing out of furnaces for relining, when many buyers expected an increase through the advent of new furnaces. Pig iron production during the first half of the year was very close to 12,500,000 tons; it was above that rate in March, and has since steadily, though but slightly, declined. Never before have American blast furnaces been subject to such steady market conditions, so as to develop the real capacity of the industry as a whole, which can now be accurately estimated at 25,000,000 tons per year; an abnormally small proportion was out for relining in March, while this midsummer the number is slightly above the normal. New capacity to come in is unusually small, until the large new plant of the United States Steel Corporation, at Gary, Ind., is heard from, about the beginning of 1908.

Market prices f. o. b. Mahoning and Shenango Valley furnaces, and making the market for Pittsburg by the addition of the 85 cents freight, are: Standard Bessemer, $17.75; basic, $17.25; No. 2 foundry, $16.50; gray forge, $15.50.

STEEL.

Bessemer and open-hearth billets are scarce, and are nominally about $27.50 for Bessemer and $28.50 for open-hearth, Pittsburg. Small billets are $30.00 and upward, and forging billets from $32.00 up. The supply of sheet bars is rather unsatisfactory, and the market is well held at $29.00, Pittsburg, for long bars and 50 cents advance for bars cut to specifications. The Carnegie Steel Company's surplus tonnage in billets and sheet bars is practically taken for the balance of the year on contracts. One cause of the scarcity of billets and sheet bars is the heavy demand for rails, requiring a larger tonnage than ever before from the interchangeable mills which roll rails, billets or sheet bars at will. The Carnegie Youngstown mill is scheduled on rails for the balance of the year, with a small proportion of billets, and no sheet bars, while at Mingo standard and small billets are being made to the almost total exclusion of sheet bars.

The Youngstown Sheet & Tube Company, which set out to build a complete Bessemer steel plant in the record time of twelve months to July 1 last, will bring its plant in about the middle of August. It is a standard two 10-ton vessel plant, with a large blooming mill on which no billets will be rolled; slabs will be taken to the company's present skelp mills, as a reheating proposition, and the blooming mill will also supply, for initial heat rolling, two new mills, a 4 x 4 billet mill and a mill to roll sheet bars and small billets. The finishing capacity is in excess of steel making capacity, and thus great flexibility is furnished. The management hopes shortly to reach 2,000 tons of ingots daily, although direct metal does not seem to be contemplated, and the effort will be watched with interest. The product for the remainder of the year seems to be disposed of.

RAILS.

The Carnegie Steel Company is now practically sold for the balance of 1906, and was the last producer to fill up. It has about 250,000 tons sold for next year. The Illinois Steel Company has about all its 1907 production sold, perhaps 750,000 tons, while the Alabama and Colorado mills have sold more than half their product for 1907. Lackawanna, Pennsylvania and Cambria are sold in less proportion. Demand for rails in the West and South has increased much more rapidly than capacity in those districts. The price on Bessemer rails being $28.00 at mill, some Western roads have been forced to place orders in Pittsburg, and thus pay a higher delivered price than had the orders been acceptable to the Chicago mill. The new open-hearth steel mill in Indiana will hardly make rails before the season of 1908, but a large portion of its prospective output could have been sold for delivery this year or next. It will be able to turn more than a million tons a year of its steel into rails if required.

FINISHED MATERIAL.

In nearly all lines mills are assured of full operation the remainder of this year, with orders now on hand and business which is certain to float in. The greatest activity in July has been in line pipe and in plates and shapes for lake vessels, nine vessels having been placed with lake shipyards in two days.

The following prices are f. o. b. Pittsburg, plus regular freight to destination:

Beams and channels, 15 inches and under, $1.70; tees, $1.75; beams and channels over 15 inches, $1.80.

Plates, tank quality, ¼ inch and heavier, to 100 inches wide, $1.60.

Sheets, 28 gauge, one pass cold rolled, $2.50.

Galvanized sheets, 28 gauge, $3.55.

Plain wire, $1.70.

Wire nails, keg, $1.85.

Tin plates, 100-pound cokes, per box, $3.75.

Fixation of Atmospheric Nitrogen.

Three interesting papers were recently presented on this problem, which is now no longer of purely theoretical interest, but has attained commercial importance both through the commercial success of the Birkeland-Enyde process in Norway (making nitrate fertilizers by means of arc discharges through air), and through the commercial success of the cyanamide process in Italy (making calcium cyanamide from calcium and nitrogen, the latter being obtained from atmospheric air by liquefaction and distillation).

Of the three papers to which we refer, two were presented by the inventors of the process just referred to. Prof. Kr. Birkeland, of the University of Christiania, read his paper before the Faraday Society in London; Prof. Ad. Frank, of Charlottanburg. lectured before the International Congress for Applied Chemistry in Rome. The paper by Prof. Birkeland shall be dealt with in this article, an abstract of the paper of Prof. Frank will be found in the Synopsis of our present issue on page 327.

The third paper, to which we referred above, was presented by Prof. Th. A. Guye before the Society of Chemical Industry in London. It was an elaborate presentation of the fundamental principles of physical chemistry on which the new industries are based; the content of the paper was essentially the same as Prof. Guye's article in our April issue, page 126, but further enlarged. We will refer to some new points of this paper in our next issue.

Prof. Birkeland's paper is very voluminous, and since the principle of the Birkeland-Eyde process and the main results have repeatedly been covered in our columns (f. i., recently in our April issue, p. 126), we will give in full only those parts of the paper which describe the equipment of the Notodden works.

The alternating-current disc-plane, which is the essential feature of the Birkeland-Eyde process, was enclosed in a special furnace, which was lined with fire-brick, and furnished with a metal casing. Diagrams, two on page 399 of our Vol. II., and on page 126 of our Vol. III., show two such furnaces, one of an older, the other of a newer type. The fire-chamber of the furnace is narrow, in the direction of the lines of force— from 5 to 15 cm. wide—made partly of perforated chamotte air being conveyed to the disc-flame, in an evenly-distributed supply, through its walls. The system of magnets is composed of two powerful electromagnets, their extremities turned in towards the fire-chamber. The magnetic circuit is closed either as in a horse-shoe magnet, or through the shield-like, cast-steel casing of the furnace. The air is driven into the central region on both sides of the flame by gentle pressure from a Root's blower; and after passing in a radial direction arrives at a peripheral channel, whence it is conducted away. The horizontal electrodes are made of copper tubing, 15 mm. in diameter; whose terminals are within about 1 cm. of one another, and are cooled by water in circulation, which keeps them from fusing. The electrodes are exchanged and repaired after being in use for about 300 hours, the exchange itself taking about 15 minutes to accomplish.

At the Notodden Saltpeter Manufactory there are three such furnaces in constant activity, each employing 500 kilowatts. These furnaces burn with an astonishing degree of steadiness, with a variation in energy of only 2 or 3 per cent, although the electrodes have no automatic regulation. It often happens that the assistant who attends to the furnaces does not need to touch any of them all through his watch; and it has happened that a furnace has burned for 40 hours without being attended to. The furnaces themselves give notice of anything going wrong, for the flame then begins to roar; and the warning comes in ample time to allow of an adjustment before the flame is extinguished. The furnaces work with a factor of reduction of 0.7. With a working potential of 5,000 volts, 3,500 volts and more are obtained on the electrodes.

As already mentioned, the furnaces are lined with fire-brick. Even the large, inner surfaces of the furnace have shown that they remain very stable, the reason of this evidently being that, in spite of the enormously high temperature of the disc-flame, the temperature on the walls does not rise above 700° C. during normal working, owing to the cooling effect of the current of air.

We have estimated that two fire-brick linings would be required per annum for each furnace; but this will certainly prove to be too high an estimate on the average, and the fire-brick furnaces will last longer than they are assumed to do. In the new saltpeter manufactory at Notodden, which is to obtain about 30,000 hp. from Svælgfos, 5 kilometers distant, it is decided that the furnaces shall be of 750 kilowatts under normal conditions, while during flood time they will be put up to 850 kilowatts. With a working tension of 5,000 volts the power factor[1] for these furnaces will be about 0.75.

We have had an opportunity of testing this type of furnace with as much as 1,000 kilowatts, but not yet for a sufficient length of time, as the alternator employed was too heavily laden.

The cost of these furnaces, including induction resistance, is at the present moment 18,000 krones, or $5,000, fully complete and capable of being immediately attached to an ordinary alternating current with a tension of 5,000 volts. The cost per kilowatt of the erection of furnaces can thus, at the present time, be put at 18 krones, or $5.00, which is, of course, surprisingly low.

There is reason to believe, however, that this low cost of erection per kilowatt will at some future time be further considerably reduced. From the experience gained up to the present, it would appear that in larger works the furnaces should be made to take larger quantities of energy than 1,000 kw. If, for instance, 2,000 kw. per furnace were absorbed, a considerable saving would be effected in the cost of erection and the working expenses.

There would be a gain in the power factor which would probably attain to 0.8, and there is no indication—if we draw conclusions by analogy from the experiments made with from 250 kw. to 750 kw. per furnace—of any loss in the output per kw.-year; on the contrary, a small gain might rather be expected there also.

Furnaces of 2,000 kw. such as these would cost, complete, between 20,000 and 22,000 krones, or about $5,500 to $6,050, including inductive resistance; so that the cost of erection would then be from 10 to 11 krones per kilowatt, an exceptionally low cost for the installation of electrical apparatus.

But it is now necessary to speak of the products of the process. The nitric oxide fumes which are formed in the furnaces and conducted away with the hot gases, fixate, after cooling, a further quantity of oxygen from the unconsumed part of the air, and turn into nitric peroxide; and this, when treated with water, combines to form nitric acid.

The volume of air thus treated at present in the Notodden Saltpeter Manufactory is 75,000 liters per minute; and as it contains only about 1 per cent of nitric oxide, it will easily be understood that in discussing the question of obtaining nitric acid from the air, it has frequently been pointed out that the great rarefaction of the gases to be worked with, greatly enhances the difficulties of the problem to be solved.

The gases that come from the furnaces with a temperature of 600° C. to 700° C. first pass through a steam boiler, the steam of which is employed in the further manufacture of the ultimate product, calcium nitrate. In the saltpeter manufactory now in course of erection, the gases will be conducted directly through the evaporation tank, an arrangement which

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[1] The author writes "factor of reduction," but evidently means what we call the power factor; i. e., the cosine of the angle of phase difference between e. m. f. and current, or the factor with which the volt-amperes must be multiplied to get the watts in an alternating-current system.

implies such an important saving of heat, that the employment of coal may presumably be avoided.

After the gases are discharged from the above-mentioned steam boiler, their temperatures is reduced to 200° C., whereupon they are conducted through a cooling apparatus to be further cooled to about 50° C. The more the gases are cooled, the more easily are they absorbed by water. The gas then enters two large oxidation chambers with acid-proof lining. Here, as already mentioned, takes place the oxidation of the compound formed in the furnaces, nitric oxide becoming nitric peroxide, which is then conducted farther into an absorption system, where the gas is converted into nitric acid.

The absorption system consists of two series of stone towers, whose internal dimensions are 2 x 2 x 10 meters, each series containing five towers, two of granite and two of sandstone, filled with pieces of quartz, over which water, and the nitric acid formed, are made to trickle, while the fifth tower in each series is filled with ordinary bricks, over which trickles milk of lime. The milk of lime quickly absorbs the rarefied nitrous gases remaining, and is converted into a compound of calcium nitrate and calcium nitrite.

The first tower yields a 50 per cent nitric acid, the second about 25 per cent, the third 15 per cent, and the fourth 5 per cent. The liquids from the fourth tower are raised by compressed air to the top of the third, those from the third to the second, and those from the second to the first, thus gradually increasing in concentration up to 50 per cent, at which density the acid is conducted into a series of open granite tanks, where it is temporarily stored.

Some of this acid is employed in the decomposition of the nitrate-nitrite combination obtained by absorption by milk of lime. By the addition of nitric acid, the nitrous anhydride contained in the nitrite is driven out, and carried back to the system of towers.

The solution resulting from this, containing pure calcium nitrate, is carried, together with the rest of the stored-up acid, into another series of granite tanks, where this mixture of acid and acetous calcium nitrate lye, reacting on ordinary limestone, is converted into a solution of neutral calcium nitrate. This neutral lye is carried farther into vaporization-chambers of iron, where it is vaporized to a boiling point of 145° C., answering to a concentration of from 75 to 80 per cent of calcium nitrate, containing about 13.5 per cent of nitrogen. This substance is then run into iron drums containing about 200 liters, where it congeals, and in that form appears on the market.

Instead of vaporizing the calcium nitrate right to a boiling point of 145°, it may be allowed to crystallize after being vaporized to a boiling point of 120° C. The crystallized calcium nitrate is treated centrifugally, when a crystalline substance is obtained as the final product, which is mainly used commercially as a fertilizer. Its composition is $Ca(NO_3)_2 + 4H_2O$, and is a salt with hygroscopic properties. In order, therefore, to make the salt more suitable for a manure it is converted into basic nitrate, which keeps dry, thus allowing of its being scattered with a sowing-machine. The manufacture of basic nitrate was suggested by Dr. Rudolph Messel, of London.

A number of manuring experiments with calcium nitrate have been made at various agricultural institutes. The results show that lime saltpeter is quite as good as the natural saltpeter, and on sandy soil even superior to it. This last fact must be ascribed to the importance of the lime contained in the saltpeter to plants in soil that is so deficient in lime.

In order to obtain a satisfactory idea of the degree of success attending the solution of the nitrogen problem, it is necessary to know approximately the cost of production of calcium nitrate produced by the process here described.

The present cost is hardly likely to be published, but there are already available various official data from which fairly definite conclusions may be drawn.

At the opening of the new Chemico-Technical Institute at the Royal Technical College in Berlin, the director of the Institute (Prof. Otto N. Witt) spoke of the utilization of atmospheric nitrogen. In doing so he gave a detailed description of the Birkeland-Eyde process, which he knew from personal experience, having been one of the experts who visited Notodden. With regard to the practical output, he stated that, according to his observation, the yield was between 500 and 600 kilogrammes of anhydrous nitric acid per kw-year.

Now, Birkeland has found that the output of the furnaces is higher than 600 kilogrammes, if it is measured by analyzing samples of gas taken out, and including in the calculation the amount of air blown through the furnaces.[1]

The measurement of large volumes of air such as those in the present case requires large, expensive, accurate gas meters. They have used a "Duplex" from the "Cie. pour la Fabrication des Compteurs et Matériel d'usine a Gaz," of Paris, which can take a maximum of about 21,000 liters per minute. By special test determinations this meter, after being mounted in a separate little building, has proved to be correct to within ½ per cent.

The reason of the considerable difference observed between the output of the furnaces and the practical output has not yet been made quite clear, and investigations into this matter are, therefore, still being persevered with.

In the estimate for the new factory of about 27,000 e. hp. at Notodden by a commission of famous foreign experts, among whom were Grandeau, Schloesing, Silvanus P. Thompson, Turrettini and Otto N. Witt, a practical output of 500 kg. of anhydrous nitric acid per kw-year has been taken for granted.

In the prospectus issued on the formation of the company that now owns the works just named, it was stated that the cost of production per ton of calcium nitrate, containing 13.2 per cent of nitrogen, would be 72.90 krones ($20.00), while the selling price per ton is put at 145.20 ($40.00), this having been reckoned from the present price of 1,100 krones (about $300) per ton of combined nitrogen.

As the building of the new factory is already far advanced, there has been ample opportunity of seeing that the cost of the works has been very correctly estimated.

In the concluding part of his paper the author discusses the theory of the oxidation of nitrogen in electric arcs.

CORRESPONDENCE.

Kryptol.

To the Editor of Electrochemical and Metallurgical Industry:

SIR:—From Mr. F. A. J. FitzGerald's letter on page 210 of your June issue, it appears that there is still some question as to the composition of "kryptol." The following facts may, therefore, be of interest:

Two samples of this material were recently received from different agents of the "Kryptol Gesellschaft" in this country.

On the assumption that "krptol" was composed of graphite mixed with various proportions of carborundum (the two samples were for use with different voltages), they were submitted to analysis and found to contain 0.23 per cent and 0.27 per cent of incombustible matter respectively. These samples of "kryptol" were, therefore, similar to those examined by Mr. FitzGerald.

I may add that the second sample contained, beside 3.80 per cent of moisture, 95.52 per cent of carbon (weighed as CO_2), hence it is obvious that the material under examination contained no admixture of sand, clay, etc.

LONDON, ENGLAND.   RICHARD SELIGMAN.

[1] Mr. Edstrom, at the St. Louis Congress (our Vol. II., page 390), stated that one kw-year, measured at the arc, yields 990 kg. of HNO_3.

Electrolytic Precipitation of Gold From Cyanide Solutions.

BY PROF. B. NEUMANN, PH. D.

The cyanide process for the extraction of gold has been in use on a large scale for about 15 years. The process was developed mainly in Transvaal, but is now employed in all gold districts. It has increased in importance to such an extent that more than one-third of the world's production of gold is now obtained by its means.

The gold is precipitated from the cyanide solutions mainly by two different methods, either by zinc precipitation or by electrolysis.

Electrolytic precipitation is generally carried on by passing the electrolyte successively through several tanks, in which a great number of lead-foil strips are suspended, being connected in parallel and used as cathodes, while iron sheets are employed as anodes. The voltage at the terminals of the path is 2 to 3 volts, the best current density about 0.5 amp. per square meter, whereby the gold is precipitated in an adherent layer on the cathodes.

In the course of time, prussian blue and iron oxide are formed at the iron anodes, whereby the electrolyte becomes impure. Andreoli proposed, therefore, to substitute lead peroxide electrodes for the iron anodes, since with the former no corrosion can take place and the baths remain pure. This proposition looks undoubtedly promising at first glance. Information on the success obtained with such anodes on a large scale is, however, scarce. According to one statement from South Africa these electrodes have not proven satisfactory, on account of impurities in the electrolyte. But this is certainly not the real reason, as is proven by some experiments of the author.

Lead electrodes were electrolytically coated with peroxide according to the method of Andreoli. In this way very thin as well as pretty thick coatings were produced, and peroxide plates made by other methods were also employed for the experiments, among them one plate which had been used for several months as anode in the electrolysis of sulphuric acid.

If such plates are used as anodes in dilute potassium cyanide solutions, the same phenomenon is always observed. After a short time, at various places, white points appear, which then change to a thick white fog, passing down in the bath and fouling it completely. The white substance is lead cyanide, formed by corrosion of the anode. This is due to the fact that all electrically-produced coatings of peroxide are porous, so that the anion passes through the pores and attacks the metallic lead below the peroxide. If, on the other hand, the peroxide coating is made very thick, cracks will occur in time even if great care is taken, and the electrolyte passes through the cracks and again reaches the metallic lead. At least the author did not succeed by any method in making peroxide anodes which would have fulfilled the requirements of practice. This explains why peroxide anodes have not been generally introduced anywhere in the electrolytic precipitation from cyanide solutions.

In accordance with the results of the author is an observation of Hamilton[1] on experiments with lead peroxide anodes in Sonora, Mexico. Concerning the construction of his electrode see the article of Hamilton. His experience is that when once the surface begins to disintegrate deterioration is comparatively rapid.

In spite of the extended use of the electrolytic precipitation process on a large scale, little reliable information is available on details of the process. The figures given for the ampere-hour efficiency of the electrolytic precipitation process differ very greatly. In a discussion of a paper of Walker, Richards[2]

remarked that in the Siemens-Schuckert process the efficiency is not more than ¼ per cent, according to experiments made in his laboratory.

On the other hand, Hamilton[3] states that in Sonora, Mexico, in runs extending over two months, the "waste of current" in one box was 87 per cent, and in another 94 per cent, so that the corresponding efficiencies were 13 and 6 per cent. It is to be remarked, however, that the solutions contained considerable quantities of silver besides gold (in sand solutions 4.54 grams gold per 100 grams silver, and in slimes solutions 1.7 gram gold with 43.5 gram silver), on the other hand, the current density used was 5.5 and 3.1 amp. per square meter respectively, which is much higher than the usual current density.

Christy[4] also uses higher current densities. According to his opinion the usual method has an efficiency of 1 to 2 per cent, and certainly not more than 5 per cent, while he tried to get an efficiency of 80 per cent by a special construction of the electrodes.

In a discussion of precipitation of gold from cyanide solutions, Sharwood[5] gives the efficiency at from 6 to 12 per cent, but is cautious enough to say that "this is said to be obtained in practice." According to his statement the precipitation of gold from 100 tons solution in 24 hours (4 grams of gold, 10 grams of silver per ton) requires 5 to 6 hp. On the basis of this statement the efficiency of the precipitation of the gold alone comes out as 2.4 per cent.

In view of these contradictory statements it did not appear superfluous to make some experiments in order to determine the ampere-hour efficiency which can be obtained in the electrolysis of dilute cyanide solutions when the current density or the concentration or the length of the run are changed.

Two runs were first made in which the current density, as usual in practice, was 0.5 amp. per square meter. In one run the solution contained 10 grams of gold per metric ton (cubic meter), the other solution contained 3 grams. These experiments were made to get some idea of the efficiency of the different tanks connected in series in practice. Later on other runs were made with lower and higher current densities.

Since the current density is very low, even small-scale experiments require rather large electrode surfaces, in order to avoid the necessity of using very small currents. In the present case a glass vessel was used of 22 cm. length, 30 cm. height and 12 cm. breadth, which contained about 8 liters of electrolyte. Thin iron sheets and thin lead sheets were used as electrodes, the dimensions being 27 by 21 cm. The electrode surface within the electrolyte was 21 by 24 cm. (i. e., 504 square centimeters on one side, or 1,008 square centimeters on both sides), the quantity of electrolyte being 6.5 liters. Two lead cathodes were always used between three iron anodes. The useful lead surface was, therefore, 2,000 square centimeters, and a current of 0.1 amp. represented a current density of 0.5 amp. per square meter.

By an electrically-driven stirrer, running at a low speed, the electrolyte was uniformly agitated. This was done since I had found that with non-uniform stirring with a water turbine the deposition of the small quantities of gold per unit of time was considerably influenced.

In order to imitate as much as possible the conditions of practice where the concentration in the different baths remains always the same, due to continuous supply of new solution, I also added in my experiments as much gold to the electrolyte in form of a concentrated potassium cyanide solution as had been precipitated by the current. In this way the quantity of potassium cyanide which had been destroyed by the current was also replaced.

The experiments became complicated, since it was not possible to simply take the cathodes after a certain time from the

[1] "Electrochemical Industry," 1904, Vol. II., pp. 131 and 372.
[2] "Electrochemical Industry," 1903, Vol. I., p. 484.
[3] "Electrochemical Industry," 1904, Vol. II., p. 134.
[4] U. S. Patent 756,328, of May 20, 1900.
[5] "Engineering and Mining Journal," 1905, p. 152.

bath and weigh them. Since each lead cathode weighed 125 to 150 grams, the result of weighing would have been inaccurate, as the quantity of gold deposited is often only a few milligrams. But a greater difficulty is due to the fact that the plates, when dry, are covered with an oxide film and with traces of lead cyanide, the weight of which may be as much as that of the deposited gold. The chief reason against the usual method of weighing and again using the same cathodes was, however, as follows:

From the experiments it is seen that in the first hours relatively small quantities of gold are deposited on the lead plates. The quantity deposited per unit of time increases with proceeding electrolysis up to a maximum. If the electrolysis is then interrupted, and if the cathode is weighed and again suspended in the bath, we do not get in the next few hours the increased efficiency which we would have obtained in the hours following the first run if no interruption had occurred, but we get only the low efficiency again which was obtained in the very first hours of electrolysis.

Thus, in an experiment 0.2200 grams of gold were deposited in 8 hours of continuous electrolysis and 0.2780 gram in 10 hours, the process having not been interrrupted. The deposit of the ninth and tenth hours amounted, therefore, to 0.0580 gram. When, however, the electrolysis was interrupted after 8 hours, the plate was weighed and again suspended in the bath, and then the electrolysis continued, the next 2 hours yielded only 0.0155 grams of gold, which quantity is very nearly that obtained in the very first 2 hours of the experiment.

This experiment shows that the lower efficiency in the beginning of the experiment is intimately connected with the surface condition of the lead cathodes. For this reason it is impossible to use again the plates after they had been weighed.

The experiment was, therefore, made in such a way that always new plates were used and were in continuous operation for the whole time. For instance, the first plates for 2 hours, the second plates for 4 hours, and the third ones for 6 hours, etc., so that an experiment on continuous electrolysis for 10 hours required really 30 hours. The quantity of gold was not determined by weighing, but by the most exact method in the dry way, by cupellation in the muffle, whereby the quantity of silver was, of course, separately determined.

The results of some such experiments are given in the following table:

Table I.

Current density=0.5 amperes per square meter; 10 grams gold per cubic meter; 0.5% potassium cvanide.

Hours.	Current, Amp.	Ampere-minutes.	Voltage.	Deposited Gold in Grams.	Ampere-hour Efficiencies in Per cents.
½	0.1	3	0.8–1.7	0.0012	0.33
1	0.1	6	1.5	0.0068	0.92
2	0.1	12	1.2–2.2	0.0264	2.06
3	0.1	18	1.9–2.1	0.0647	2.93
4	0.1	24	1.9–2.2	0.1027	3.49
5	0.1	30	1.7	0.1403	3.81
6	0.1	36	1.9–2.0	0.1661	3.77
8	0.1	48	1.7–1.7	0.2089	3.55
14	0.1	84	1.7–1.9	0.3262	3.17

The figures for the efficiency refer to the whole time. They show that the efficiency becomes a maximum at a certain time and decreases again later on. At the end of this experiment 0.025 per cent free potassium cyanide was still in the solution. Quite some potassium cyanide had been destroyed during the experiment by electrolysis and air. The voltage increases with decreasing percentage of potassium cyanide.

Table II.

Current density=0.5 amp. per square meter; 3 grams gold per cubic meter.

Hours.	Current Ampere.	Ampere-minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
1	0.1	6	0.6–1.9	0.0005	0.007
2	0.1	12	1.9	0.0101	0.69
3	0.1	18	1.85	0.0195	0.88
4	0.1	24	2.0	0.0307	1.08
6	0.1	36	2.1	0.0559	1.26
8	0.1	48	2.1	0.0781	1.33
12	0.1	72	1.9–2.0	0.1160	1.31

The results of both tables show that a maximum of efficiency is obtained after some time, and that even with constant concentration of gold the efficiency decreases again slowly when the run is continued for a longer time. The experiments also show that by employing current densities, such as used in practice, the ampere-hour efficiency is surely not more than 4 per cent, even with rich solutions, but that on the contrary in longer runs, and by using the same cathodes for several weeks, the ampere-hour efficiency will be in the average considerably below this figure. In electrolyzing poorer solutions, such as in Table II., the maximum of efficiency reached is not more than 1.33 per cent. The total average of the efficiency in a plant with several cells in series will, therefore, not be more than, say, 2 per cent, but will remain considerably below 1 per cent if the run is continued for a long time, as was also found by us by special experiments.

It is evident that with smaller current densities a better current efficiency could be obtained. For practical purposes, however, the current then becomes too small; i. e., the deposit per unit of time is too small. The solutions must be passed through the cells at a lower speed or it becomes necessary to connect more cells in series.

To get more information on these points two runs were made, the results of which are given in Tables III. and IV.

Table III.

Current density=0.25 amp. per square meter; 10 grams gold per cubic meter; 0.05% of potassium cyanide.

Hours.	Amperes.	Ampere minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
2	0.05	6	0.5	0.0049	0.67
4	0.05	12	1.5	0.0485	3.30
6	0.05	18	1.4	0.1270	5.75
8	0.05	24	1.3	0.2200	7.48
10	0.05	30	1.20	0.2780	7.56
12	0.05	36	1.15	0.3267	7.40
14	0.05	42	1.05	0.3737	7.25

Table IV.

Current density=0.25 amp. per square meter; 3 grams gold per cubic meter; 0.05% of potassium cyanide.

Hours.	Amperes.	Ampere-minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
4	0.05	12	1.7	0.0056	0.38
6	0.05	18	1.7	0.0341	1.54
8	0.05	24	1.6	0.0666	2.26
10	0.05	30	1.55	0.1046	2.84
12	0.05	36	1.50	0.1346	3.05
14	0.05	42	1.40	0.1624	3.15

A glance at the Tables III. and IV. shows that with smaller current densities considerably better results were obtained than in the first two runs. In the richer solution the maximum efficiency was 7.56 per cent, in the poorer solution somewhat more than 3 per cent.

The gradual decrease of the voltage in Tables III. and IV. is explained by the fact that the greater deposition of gold requires replacing greater quantities of gold, whereby more potassium cyanide was introduced into the solution than was destroyed by the current in the same time.

We now give several runs in which higher current densities than in Table I. and II. were used.

TABLE V.

Current density=2.4 amp. per square meter; 10 grams gold per cubic meter; 0.07% of potassium cyanide.

Hours.	Minutes.	Amperes.	Ampere-minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
..	20	0.12	2.4	1.8–2.2	0.0004	0.13
1	15	0.12	9.0	2.2–2.3	0.0045	0.41
2	..	0.12	14.4	1.8–2.2	0.0064	0.36
3	30	0.12	25.2	2.1–2.2	0.0079	0.26
7	..	0.12	56.4	2.2	0.0084	0.12

TABLE VI.

Current density=4 amp. per square meter; 10 grams gold per cubic meter; 0.07% of potassium cyanide.

Hours.	Minutes.	Amperes.	Ampere-minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
..	5	0.20	1	2.5	0.0001	0.08
..	20	0.20	4	2.5	0.0004	0.08
..	40	0.20	8	2.5	0.0015	0.16
1	..	0.20	12	2.5	0.0036	0.24
2	..	0.20	24	2.5	0.0058	0.20
3	..	0.20	36	2.5	0.0066	0.15
4	..	0.20	48	2.5	0.0068	0.11
6	..	0.20	72	2.5	0.0073	0.08

FIG. I.—EFFECT OF CURRENT DENSITY ON DEPOSITION OF GOLD.

TABLE VII.

Current density=9 amp. per square meter; 10 grams gold per cubic meter; 0.07% of potassium cyanide.

Hours.	Minutes.	Amperes.	Ampere-minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
..	15	0.45	9.	2.8–3	0.0004	0.05
..	40	0.45	18.	3.0	0.0035	0.16
1	10	0.45	31.5	2.8–3	0.0051	0.13
3	5	0.45	83.3	3.0	0.0062	0.06
5	15	0.45	141.0	3.0	0.0069	0.04

The runs V., VI. and VII. were made in a smaller vessel containing 1.25 liters of gold solution with a lead cathode surface of 500 square centimeters. The electrolyte was agitated by means of a stirrer. The temperature was the ordinary temperature of the room. Gold was always present in excess, but was not regularly replaced.

The results show that with higher current densities the ampere-hour efficiency becomes very poor, and do not even reach ½ per cent.

If we give a summary of the *best* results obtained in the different runs with the same concentrations we get the following:

At 0.25 amp. per sq. meter, the max. amp.-hour efficiency was 7.56%
" 0.50 "   "   "   "   "   "   "   " 3.81%
" 2.40 "   "   "   "   "   "   "   " 0.41%
" 4.00 "   "   "   "   "   "   "   " 0.24%
" 9.00 "   "   "   "   "   "   "   " 0.16%

In a long run with 2.5 amps. per square meter, extending over a month and a half (with 9 grams of gold per cubic

FIG. 2.—EFFECT OF CURRENT DENSITY ON DEPOSITION OF GOLD.

meter, and 0.05 per cent potassium cyanide), the average amprere-hour efficiency was 0.12 per cent, the voltage being 2.3 to 2.6.

The average ampere-hour efficiency in the electrolytic precipitation of gold from cyanide solutions on a large scale in runs extending over a longer time is, therefore, usually below 1 per cent.

In the adjoining diagrams the results are plotted graphically. Fig. 1 shows the increase of the deposition of gold under the different conditions in the course of time. Fig. 2 shows the same results on a larger scale for the runs V. to VII. In Fig. 3 the ampere-hour efficiency is plotted as obtained during the progress of electrolysis in the different solutions with the different current densities.

* * * * *

In the Siemens & Halske process strips of lead foil are used as cathodes. Lead has the advantage that the whole electrode with its gold deposit may be molten, and the gold then be easily separated from the lead.

Pfleger proposed to use fabrics of lead wire in screen form through which the electrolyte was intended to flow. This modification has not been introduced into the practice.

Andreoli proposed iron and aluminium for the cathode, and suggested to plunge the cathodes with their gold deposits into a molten lead bath which would alloy with the gold. Experiments in the laboratory gave good results, but on a large scale the method failed.

While, as stated, very low current densities are used in prac-

FIG. 3.—EFFECT OF CURRENT DENSITY AND CONCENTRATION ON EFFICIENCY.

tice in order to get a firmly adherent deposit of gold, there are other cases where high current densities are intentionally employed. Hamilton[6] states that at Sonora current densities of about 2.5 amps. per square meter are used with the intention of depositing the gold in the form of a fine slime, which

[6] "Electrochemical Industry," Vol. 11, page 131.

is then treated in the same way as the similar product from the zinc process. The lead foil was brushed off every day or two, but the lead foil suffered thereby. Tin plate instead of lead foil proved very successful later on.

Various inventors have proposed to use mercury as cathode (Locoy, Eltonhead, Stewart and others). The separation of the precious metals from the mercury could be performed in a very simple way. The same idea is at the bottom of the construction of those apparatus in which leaching and precipitation of gold are carried out in one and the same receptacle (Riecken process, Pelatan-Clerici process, Aurex-Sluice, etc.). In the latter forms of apparatus mercury absorbs relatively little gold, but it may be assumed that a moved mercury mass could alloy with quite considerable quantities of gold, so that the mercury cathode would show certain advantages over a lead cathode which can remain in the bath for one or two months only.

However, von Gernet[7] has shown that mercury electrodes are not practicable on a large scale. For instance, in order to precipitate within 24 hours the gold from 100 tons of cyanide solutions containing 5 dwt. (7.775 grams) per ton, a mercury surface of 10,000 square feet (900 square meters) would be required. In order to cover the whole bottom of the cell the mercury layer would have to be at least ¼ inch (0.6 centimeter) thick; therefore at least 250 cubic feet (5.4 cubic meters) of mercury would be required, that is, about 75 tons. In this calculation, von Gernet evidently has assumed double the usual current density. The cost of such a quantity of mercury alone would prevent the application of mercury as cathode even if we do not consider the losses of mercury during operation.

A proposition of Christy[8] is to the effect to deposit the precious metals from cyanide solutions in the usual way upon iron or lead cathodes, and to use these electrodes when covered with gold afterwards as anodes in a stronger cyanide solution, and to dissolve the gold from them and redeposit it on gold cathodes. This proposition has been claimed to be impracticable by von Uslar and Erlwein[9]. First, because too much dead capital is required in form of gold cathodes, and, secondly, because it would be difficult to control the process so that there are no losses of current or electrode material.

It is true, Christy's proposition has some weak points, but certainly it does not fail for the reason given by von Uslar and Erlwein. Since electrolytic refining of gold is used on a rather large scale in practice, it is proven that the quantity of gold held back in form of cathodes in the cells is no economical obstacle to carrying out the process. On the other hand, with such an expensive metal as gold a low ampere-hour efficiency is of not very great importance, especially as gold is monovalent in cyanide solutions, so that 7.356 grams are deposited per ampere-hour.

The economical question is, whether the cost of the treatment of the lead cathodes, with their gold deposit, is cheaper than the electrolytic dissolution of gold from an anode and redeposition on a gold cathode.

At the suggestion of Prof. Dieffenbach, experiments have been made in this laboratory which are based on a somewhat similar idea. We intended to combine the transportation of the gold from anode to cathode with simultaneous refining. If the desired purpose was to be attained it was necessary that the layer of gold should be easily dissolved from the anode (which had first served as cathodes) without the electrodes being attacked or destroyed in any way when being used.

The first question to be answered is whether the lead electrodes generally used as cathodes for gold deposition are suitable for this purpose. Hydrochloric acid and chlorides cannot be used in this case as electrolytes for the subsequent refining action, since lead is attacked by them when used as

[7] Proceedings Chem. and Metallurgical Soc. of South Africa, Vol. I.
[8] U. S. patent 643,096.
[9] Monographien f. angewandte Elektrochemie, Vol. VII.

anode. Potassium cyanide solutions are also unsuitable, as has been established by experiments, since lead cyanide is formed at the anodes, which makes the electrodes unsuitable for further use. The proposition of Christy of using lead electrodes in potassium cyanide solutions for this purpose is, therefore, impracticable. The same must be said concerning the iron anodes recommended by him, since iron oxide and prussian blue are formed anodically, which render impossible the deposition of gold on such plates when used as cathodes later on. Lead and iron are, therefore, impractical for the desired double purpose.

With respect to the use of carbon, various practical men have stated that carbon cannot be employed well as anode in potassium cyanide solutions, since it is soon destroyed. However, the amount of destruction certainly depends much on the nature of the carbon. On the other hand, carbon, when used as cathode in a potassium cyanide solution, should be quite durable, since it is covered in a very short time with a layer of gold. As a matter of fact a carbon electrode, when used in a run extending over one and a half months with very dilute potassium cyanide solutions, proved quite suitable. Acheson graphite is excellent for this purpose. Gold is deposited on carbon quite uniformly and densely, and much higher current densities may be used than are otherwise employed.

The following experiments were made by Mr. Johnny Johnson:

Carbon plates were first covered with gold. For this purpose the carbon cathodes were placed in a glass vessel containing an electrolyte with 0.0269 gram of gold and 1.52 potassium cyanide per liter. Electrolysis was carried out for 15 hours with a current of 0.15 to 0.2 amp., the electrode surface being 225 square centimeters, the voltage 2. The gold was deposited in adhering form.

When these electrodes had thus been covered with gold they were placed as anodes in a glass vessel containing 1½ liters of rather concentrated potassium cyanide solution. Gold-plated copper sheets were used as cathodes. Table VII. shows the results obtained in eight experiments with different current densities.

TABLE VII.

Hours.	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Amperes per Square Meter.	KCN per Liter.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cent.
¼	0.496	0.1465	5.0	992	177.6	......	....
¼	0.329	0.0972	5.0	659	177.6	......	....
1	0.325	0.3789	11.0	296	135.	......	....
1	0.334	0.3959	28.3	118	135.	......	....
16	0.200	......	28.8	69	135.	0.0065	0.03
3½	0.164	0.6782	28.8	57	121.	0.0062	0.15
3	0.084	0.3034	28.8	30	121.	0.0065	0.34
4	0.070	0.3286	28.8	24	121.	0.0092	0.45

From this table it is seen that at high-current densities no gold whatever was deposited, but even at lower current densities the quantity of the deposit and the ampere-hour efficiency were so low that the use of potassium cyanide solutions for the purpose of transmitting gold from the anode to the cathode is practically impossible.

The gold dissolved from the anode very quickly. In the last experiments the gold dissolved in such a loose form that it could be rubbed off with the finger. The experiments 3 to 5 were made with an electrolyte which contained gold at the start, but these experiments do not show any difference from the others.

For the transportation of the gold from the anode to the cathode we then used gold chloride solutions instead of potas-

sium cyanide solutions, whereby the experience of Wohlwill[19] on gold refining could be made use of.

In order to accelerate the deposition of the gold on the carbon plates we used a gold chloride solution containing free hydrochloric acid as electrolyte instead of potassium cyanide. A gold-plated carbon or platinum electrode was the anode. The electrolyte was heated to 60° or 70° C., and was agitated by means of a stirring device. The cathode surface in contact with the electrolyte was 10 square centimeters, the current 0.4 amp., the current density, therefore, 400 amps. per square meter. The gold was deposited with a good ampere-hour efficiency in such adhesive form that a strong stream of water did not tear off anything. The deposit was smooth in the beginning, but later on warts appeared.

The same apparatus was used for transporting the gold from the anode to the cathode. Fig. 4 shows the arrangement of the experiments. B is the storage battery, Z the cell with stirrer, V a voltmeter, A an ammeter, K a copper voltameter. R a regulating resistance and W a water rheostat.

The temperature of the electrolyte was maintained at 60° to 75° C. The anodic current density was 1,000 to 1,500 amps. per square meter. The cathodic current density varied in the different experiments. A platinum plate was used as cathode.

Several series of tests were carried out.

A. EXPERIMENTS WITH DILUTE SOLUTIONS.

The electrolyte consisted of an aqueous solution of gold chloride with 3 per cent free hydrochloric acid. The content of gold varied somewhat. The current density was varied between 140 and 1,000 amps. per square meter. The voltage at the terminals was 0.2 to 0.4.

TABLE IX.

Hours.	Minutes.	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Amperes per Square Meter.	Gold per Liter of Electrolyte. before & after Experiment.		Deposited Gold in Grams.	Calculated Ampere-hour Efficiency in Per Cent.
..	10	0.33	......	5.00	1000	22.05	....	....	.....
..	15	0.53	0.0962	6.25	850	4.45	..	.....	.....
1	30	0.44	0.7828	8.75	503	4.70	1.15	2.3430	145.0
..	54	0.42	0.4450	10.00	418	4.75	3.45	1.3748	150.0
..	45	0.31	0.2796	10.00	310	7.25	6.10	0.9842	170.0
1	20	0.23	0.3668	10.00	233	3.00	2.60	1.3243	175.0
1	35	0.14	0.2654	10.00	141	3.30	2.85	1.0806	198.0

In the first two experiments of this table the electrolysis was soon interrupted, since the gold was deposited as a dark green slime at the cathode. In the third experiment it also adhered very poorly. The lower the current density the better became the adhesiveness of the gold to the cathode and the lighter became its color.

The explanation of the abnormally high figures for the current density is that in calculating the same the gold was assumed to exist in the chloride solutions as a three-valent ion, while as a matter of fact part of it migrates as monovalent ion, as will be discussed later.

The ampere-hour efficiency becomes also better with decreasing current density. It is also remarkable that the solution was always poorer in gold after the experiment than before. That means that more gold is deposited on the cathode than is dissolved from the anode, and the loss of gold from the electrolyte is greater at higher current densities than at medium ones.

Experiments were then made with a similar electrolyte, to which, however, common salt was added to increase the conductivity.

[19] Wohlwill, Zeit. f. Elektrochemie, 1897-8. Vol. IV., page 381; see also Tuttle, "Electrochemical Industry," Vol. I., page 157, and Wohlwill, "Electrochemical Industry," Vol. II., pages 221 and 261.

B. EXPERIMENTS WITH DILUTE SOLUTIONS WITH AN ADDITION OF SODIUM CHLORIDE.

The apparatus was the same as before. The electrolyte contained 30 grams of hydrochloric acid per liter, 200 grams of sodium chloride and 7.5 grams of gold. On account of the better conductivity the voltage was somewhat lower than before. The conditions of the experiments and the results may be seen from the following Table X:

TABLE X.

Hours.	Minutes.	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Current Density in Amperes per Square Meter.	Gold per Liter of Electrolyte. before & after Experiment.		Deposited Gold in Grams.	Calculated Ampere-hour Efficiency in Per Cent.
..	45	0.498	0.4411	10.0	498	7.5	3.00	0.9988	109
.	35	0.424	0.2926	10.0	424	7.5	3.50	0.7068	117
..	40	0.324	0.2559	10.0	324	7.5	3.85	0.6842	129
..	45	0.221	0.1962	10.0	221	7.5	4.25	0.5673	140
1	10	0.134	0.1848	10.0	134	7.5	4.48	0.5621	147

The deposits of gold in this experiment were much more solid than in the former experiments, especially at lower cur-
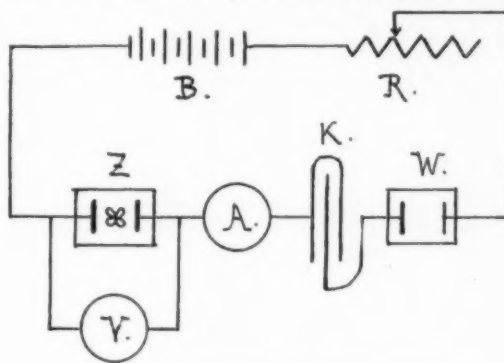

FIG. 4.—ARRANGEMENT OF EXPERIMENTS.

rent densities. The color of the deposit was a light yellow. The quantities of gold deposited have become less, and the loss of gold from the electrolyte is less than before. If little gold is only left on the anode, chlorine is also set free.

Some experiments were then made in which some Kieselguhr was added to the chlorine solutions containing free acid. The object was to remove the hydrogen bubbles and to increase the density of the deposit.

C. EXPERIMENTS WITH GOLD CHLORIDE SOLUTIONS CONTAINING HYDROCHLORIC ACID WITH KIESELGUHR.

A solution was first made up of the same composition as in the experiments of series A. Kieselguhr slime was then added. The addition amounted to 1 gram of Kieselguhr per 150 cubic centimeters of solution. The more coarse particles of Kieselguhr removed not only the hydrogen bubbles but also some of the gold deposit from the cathode. The following experiments were made with the finer particles of Kieselguhr:

TABLE XI.

Hours.	Minutes.	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Current Density in Amperes per Square Meter.	Gold per Liter of Electrolyte. before & after Experiment.		Deposited Gold in Grams.	Calculated Ampere-hour Efficiency in Per Cent.
..	50	0.514	0.5057	10.0	514	7.5	3.20	1.4142	135.0
..	55	0.394	0.4275	10.0	394	7.5	3.90	1.2765	144.5
1	7	0.309	0.4080	10.0	309	7.5	4.05	1.2251	145.4
1	15	0.214	0.3161	10.0	214	7.5	4.22	0.9929	152.0
2	25	0.095	0.2714	10.0	95	7.5	4.40	0.9802	174.5

The condition of the deposits was better than in the experiments of series A, but the deposits were not as solid and dense as in series B. As before, the deposits improved with decreasing current density. Evidently small particles of gold were also torn off by the Kieselguhr, so that the efficiency is somewhat lower than with the solutions of the same composition in series A. The deposit of gold also contained some Kieselguhr.

Since, according to Wohlwill, an electrolyte which contains 25 to 30 grams of gold per liter is best suited for gold refining, some experiments were made with more concentrated gold solutions. One experiment was made without any addition, the other with the addition of common salt to increase the conductivity.

D. Experiments With Concentrated Gold Solutions.

The electrolyte contained 23.3 grams of gold and 30 grams of hydrochloric acid per liter in the first experiment, and 29.15 grams of gold, 30 grams of hydrochloric acid and 200 grams of sodium chloride in the second experiment. The voltage was 0.3 in the first case and 0.2 in the second case.

TABLE XII.

Hours.	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Current Density in Amperes per Square Meter.	Gold per Liter of Electrolyte.		Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cent.
					before & after Experiment.			
1	0.344	0.4058	10.0	344	23.3	22.4	1.1508	137
1	0.352	0.4156	10.0	352	29.15	27.8	1.1605	135

The content of gold in the solution decreased little in comparison with the concentration, but the absolute quantity which disappeared from the solution was about the same as before.

FIG. 5.—EFFICIENCY AS FUNCTION OF CURRENT DENSITY.

The ampere - hour efficiency in the last experiment is the same as in the corresponding experiment with dilute solutions. In the preceding experiment, however, the ampere-hour efficiency is much lower than in the corresponding experiment with dilute solutions. The deposits from concentrated solutions are considerably more dense and solid than those from dilute solutions. Especially the last deposit was excellent in various respects of quality.

By plotting the results of the different series of experiments graphically, the diagram of Fig. 5 is obtained. The results of series A and B only were plotted, since series C showed the same character but exhibited some irregularities on account of the content of Kieselguhr. The curves show clearly how the ampere-hour efficiency decreases in all electrolytes with increasing current density.

A few words may now be said concerning the abnormally high ampere-hour efficiencies. These varied in the experiments between 109 and 198 per cent, on the usual assumption that gold is three-valent in a gold chloride solution. But the so-called gold chloride solutions do not contain the gold as $AuCl_3$, but in form[11] of $HAuCl_4$. This compound, as well as its alkali salts, are complex salts, which first decompose into H and $AuCl_4$.

$AuCl_4$ easily gives off Cl, and the resulting $AuCl_3$ is then decomposed into its constituents.

Wohlwill[12] has further shown that from a gold anode in presence of chlorine gold goes anodically into the solution only if the conditions are fulfilled (HCl, NaCl) that the complex compound $HAuCl_4$ or $NaAuCl_4$ (that is the anion $AuCl_4$) can be formed. The gold is, therefore, deposited on the cathode by a secondary reaction in any case.

By the use of a voltameter Wohlwill has found in his investigations that more gold is always deposited on the cathode than corresponds to the equivalent Au/3. Without any doubt gold dissolves anodically also as AuCl, just as from a copper anode cuprous ions are formed besides cupric ions.

Wohlwill has proven that this AuCl is again decomposed into gold and $AuCl_3$; $3AuCl = AuCl_3 + 2Au$ (just as Cu_2SO_4 is decomposed into $CuSO_4$ and Cu).

This decomposition of AuCl is, however, not complete in the neighborhood of the anode, and a part of the monovalent gold ions reaches the cathode and is there discharged. The number of monovalent gold ions passing from the anode into solution depends greatly on the current density, as is also shown by our experiments. With the highest current densities the formation of monovalent gold ions is a minimum, at the lowest current density it is a maximum. Therefore, the excess of gold deposited above the theoretical value on the assumed three-valent basis is a maximum in the latter case.

The phenomena of solution at the anode have been discussed in detail by Wohlwill, and there is no further need of referring here to this subject. From Wohlwill's experiments it follows that the "anodic solution equivalent" of gold is higher than its "cathodic deposition equivalent," that is, the loss of weight of the anode exceeds considerably the increase of weight of the cathode.

The question now comes up whether this method of transporting gold from an anodic carbon plate to a gold foil cathode, with a gold chloride solution for refining purposes, can be carried out in practice. This question must certainly be answered in the affirmative. With a potassium cyanide solution, however, refining would not be possible, but it is practical with a chloride solution containing free acid. The cyanide solutions often contain large quantities of silver besides gold, also some copper, lead and iron. Of course, the same metals are then also found in the raw gold, obtained by electrolytic precipitation from the cyanide solutions. In refining this raw gold with a chloride solution only gold is deposited at low voltages, while silver is precipitated as silver chloride; copper and iron pass into solution, but are not deposited, while lead is oxidized and passes into the anode slime. With such a refining process very pure electrolytic gold is, therefore, obtained.

To sum up the results of this investigation we have found:
That for the electrolysis of dilute cyanide solutions the lead peroxide anodes recommended by Andreoli are unsuitable;

That in the electrolysis of such gold solutions the ampere-hour efficiency is exceedingly low and will remain in the average far below 1 per cent;

That carbon electrodes of a certain quality are quite durable in dilute cyanide solutions, and may be used under certain conditions in the place of lead cathodes;

That the use of carbon electrodes becomes necessary if it is intended that the raw gold deposited on the same from the cyanide solution should be directly refined in a succeeding electrolysis, which can be done in a gold chloride solution containing free acid.

In gold chloride solutions, to which sodium chloride is added to improve the conductivity, exceedingly high ampere-hour efficiencies may be obtained with low-current densities.

Such a process would make unnecessary the whole treatment of the lead cathodes, which can alloy only with 2 to 12 per cent of gold, and the final product would be fine gold of about 0.988 purity instead of raw gold of 0.800 or 0.900 purity. Electrochemical Laboratory, Institute of Technology, Darmstadt, Germany.

[11] Hittorf, Poggendorff's Ann., 1859, Vol. 106, page 522.

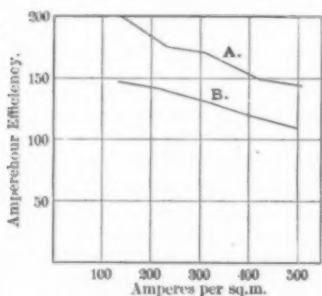[12] Zeit. f. Elektrochemie. 1897-8, Vol. IV., page 381.

Standard Determination of Carbon and Sulphur.

As was already briefly mentioned in our last issue the Committee on standard methods for the analysis of iron of the American Foundrymen's Association recommended in a report at the Cleveland meeting the following methods of determining total carbon and sulphur:

CARBON DETERMINATION.

The train shall consist of a preheating furnace, containing copper oxide (Option No. 1), followed by caustic potash (1.20 S. Gr.), then calcium chloride, following which shall be the combustion furnace, in which either a porcelain or platinum tube may be used (Option No. 2). The tube shall contain 4 or 5 inches of copper oxide between plugs of platinum gauze, the plug to the rear of the tube to be about the point where the tube extends from the furnace. A roll of silver foil about 2 inches long shall be placed in the tube after the last plug of platinum gauze. The train after the combustion tube shall be anhydrous cupric sulphate, anhydrous cuprous chloride, calcium chloride, and the absorption bulb of potassium hydrate (Sp. Gr. 1.27), with prolong filled with calcium chloride (Option No. 3). A calcium chloride tube attached to the aspirator bottle shall be connected to the prolong.

In this method a single potash bulb shall be used, a second bulb, as sometimes used for a counterpoise being more liable to introduce error than correct any error in weight of the bulb in use, due to change of temperature or moisture in the atmosphere.

The operation shall be as follows: To 1 gram of well-mixed drillings add 100 c. cm. of potassium copper chloride solution and 75 c. cm. of hydrochloric acid (conc.). As soon as dissolved, as shown by the disappearance of all copper, filter on previously washed and ignited asbestos. Wash thoroughly the beaker in which the solution was made with 20 c. cm. of dilute hydrochloric acid (1-1); pour this on the filter and wash the carbon out of the beaker by means of a wash bottle containing dilute hydrochloric acid (1-1), and then wash with warm water until all the acid is washed out of the filter. Dry the carbon at a temperature between 95° and 100° C.

Before using the apparatus a blank shall be run and if the bulb does not gain in weight more than 0.5 mg., put the dried filler into the ignition tube and heat the preheating furnace and the part of the combustion furnace containing the copper oxide. After this is heated start the aspiration of oxygen or air at the rate of three bubbles per second, to show in the potash bulb. Continue slowly heating the combination tube by turning on two burners at a time, and continue the combustion for 30 minutes if air is used; 20 minutes if oxygen is used. (The Shimer crucible is to be heated with a blast lamp for the same length of time.)

When the ignition is finished turn off the gas supply gradually, so as to allow the combustion tube to cool off slowly, and then shut off the oxygen supply and aspirate with air for 10 minutes. Detach the potash bulb and prolong, close the ends with rubber caps and allow it to stand for 5 minutes then weigh. The increase in weight multiplied by 0.27273 equals the percentage of carbon.

The potassium copper chloride shall be made by dissolving 1 pound of the salt in 1 liter of water and filtering through an asbestos filter.

Option No. 1.—While a preheater is greatly to be desired, as only a small percentage of laboratories at present use them, it was decided not to make the use of one essential to this method—subtraction of the weight of the blank to a great extent eliminating any error which might arise from not using a preheater.

Option No. 2.—The Shimer and similar crucibles are largely used as combustion furnaces, and for this reason it was decided to make optional the use of either the tube furnace or one of the standard crucibles. In case the crucible is used it shall be followed by a copper tube 3-16 inch inside diameter and 10 inches long, with its ends cooled by water jackets. In the center of the tube shall be placed a disc of platinum gauze, and for 3 or 4 inches on the side toward the crucible shall be silver foil, and for the same distance on the other side shall be copper oxide. The ends shall be plugged with glass wool, and the tube heated with a fish tail burner before the aspiration of air is started.

Option No. 3.—Barium hydrate is so extensively used as an absorbent for the carbon dioxide that your committee decided to make no recommendation as to whether its use should be made optional, but leave that point to be decided at the general meeting of the Metallurgical Section.

SULPHUR DETERMINATION.

Dissolve slowly a 3-gram sample of drillings in concentrated nitric acid in a platinum dish covered with an inverted watch glass. After the iron is completely dissolved add 2 grams of potassium nitrate, evaporate to dryness and ignite over an alcohol lamp at red heat. Add 50 c. cm. of a 1 per cent solution of sodium carbonate, boil for a few minutes, filter, using a little paper pulp in the filter if desired, and wash with a hot 1 per cent sodium carbonate solution. Acidify the filtrate with hydrochloric acid, evaporate to dryness, take up with 50 c. cm. of water and 2 c. cm. of concentrated hydrochloric acid, filter, wash, and after diluting the filtrate to about 100 c. cm. precipitate with barium chloride, filter, wash well with hot water, ignite and weigh as barium sulphate, which contains 13.733 per cent of sulphur.

Metallurgical Calculations.

BY J. W. RICHARDS, PH. D.
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HEAT BALANCE SHEET OF BLAST FURNACES.

Problem 54.

(Data partly from paper by the author, Transactions American Institute of Mining Engineers, 1905).

A blast furnace running on Lake Superior ore has the following charges, per 100 of pig iron produced:

Hematite ore: 177.6; composition	H_2O	— 10.0	per cent.
	SiO_2	— 10.0	"
	Al_2O_3	— 3.5	"
	Fe_2O_3	— 76.5	"
Limestone: 44.4; composition	SiO_2	— 5.0	"
	MgO	— 4.8	"
	CaO	— 47.6	"
	CO_2	— 42.6	"
Coke: 95.8; composition	SiO_2	— 5.3	"
	CaO	— 5.3	"
	H_2O	— 1.0	"
	C	— 88.0	"
Pig iron produced: 100; composition	Si	— 1.0	"
	C	— 4.0	"
	Fe	— 95.0	"
Gases produced: composition dry:	CO_2	— 13.0	"
	CO	— 22.3	"
	N_2	— 64.7	"

Blast used: Contained 5.66 grains of moisture per cubic foot of dry air, at 24° C. = 75° F.

Charges in pounds: Coke, 10,200; ore, 20,000; stone, 5,000.

Product per day: Pig iron, 358 tons = 801,920 pounds.

Coke used per day: 768,626 pounds.

Temperature of blast 720° F. = 382° C.

Temperature of waste gases = 538° F. = 281° C.

Displacement of blowing engines = 40,000 ft³ per min.

Heat in one unit of pig iron = 325 Calories.

Heat in one unit of slag = 525 Calories.

Cooling water, per day, heated 50° C. = 300,000 gallons.

Required: (1) The volume of gases per 100 kg. of pig iron.

(2) A balance sheet of materials entering and leaving the furnace, per 100 units of pig iron.

(3) The volume and weight of blast per 100 kg. of pig iron.

(4) The efficiency of the blowing plant.

(5) The heat balance sheet of the furnace.

(6) The proportion of the fixed carbon of the fuel burnt at the tuyeres.

(7) The proportion of the whole heat generated at the tuyeres.

(8) The proportion of the iron reduced in the furnace which is reduced by solid carbon from FeO.

(9) The theoretical maximum of temperature at the tuyeres.

(10) The theoretical maximum if all the moisture were removed from the blast.

Solution: (1) To find the volume of the gases:

Carbon in the coke used 95.8×0.88 $= 84.3$ kg.

Carbon in the limestone $44.4 \times 0.426 \times \dfrac{3}{11} =$ 5.2 "

Total carbon entering furnace $= 89.5$ "
Carbon in 100 of pig iron $= 4.0$ "

Carbon entering gases $= 85.5$ "

Carbonous and carbonic oxides in gases $= 35.3$ per cent.
Carbon in 1 m^3 of dried gas $= 0.54 \times 0.353 = 0.19062$ kg.

Dry gas per 100 kg. of pig iron $= \dfrac{85.5}{0.19062} = $ **448.5** m^3. (1)

Dry gas per 100 oz. of pig iron $= 448.5$ $ft.^3$
Dry gas per 100 pounds of pig iron $=$
448.5×16 $= 7176.0$ "
Dry gas per 2240 pounds of pig iron $=$
7176.0×22.4 $= 160,742$ "

Dry gas per minute $= \dfrac{160,742 \times 358 \ (tons)}{60 \times 24} = 39,962$ "

(2) BALANCE SHEET OF MATERIALS, PER 100 OF PIG-IRON

	Charges		Pig Iron		Slag		Gases	
Ore 177.6	Fe^2O^3	135.6	Fe	95.0	......		O	40.7
	H^2O	17.8			......		H^2O	7.8
	SiO^3	17.8	Si	1.0	SiO^3	15.7	O	1.1
Flux 44.4	SiO^2	2.2	......		SiO^2	2.2	......	
	CaO	21.1	......		CaO	21.1	......	
	MgO	2.1	......		MgO	2.1	......	
	CO^2	19.0	......		......		CO^2	19.0
Fuel 95.8	C	84.3	C	4.0	......		C	80.3
	SiO^2	5.3	......		SiO^2	5.3	......	
	CaO	5.3	......		CaO	5.3	......	
	H^2O	0.9	......		......		H^2O	0.9
Blast 422.2	O^2	96.4	......		......		O	96.4
	N^2	321.3	......		......		N^2	321.3
	H^2O	4.5	......		......		} H	0.5
							} O	4.0
	Total 740.0		100.0		58.0		582.0	

The charges are calculated simply from the weights of ore, flux and fuel used, and their percentage composition. The blast is calculated as given in solution of requirement (3).

The 1.0 of silicon in the pig iron requires $1.0 \times (60 \div 28) = 2.1$ parts of SiO^2 to furnish it, leaving 15.7 of unreduced SiO^2 to go into the slag, and 1.1 of oxygen to the gases.

(3) The balance sheet shows that the solid charges furnish to the gases 41.8 of O, 19.0 of CO^2 and 18.7 of H^2O. The H^2O goes as such into the gases, and therefore its oxygen is not present in the sample of dry gas analysed. The oxygen in CO^2 is $19.0 \times (32 \div 44) = 13.8$ kg., which added to the 41.8 gives 55.6 of oxygen getting into the gases as CO or CO^2, from the solid charges.

The oxygen in the CO and CO^2 of the gases is to be calculated from the oxygen in unit volume of gas and the total volume of gas produced. The oxygen in 1 cubic meter of gas will be:

O in CO $= 0.223 \times \left(0.09 \times \dfrac{28}{2}\right) \times \dfrac{16}{28} = 0.16056$ kg.

O in $CO^2 = 0.130 \times \left(0.09 \times \dfrac{44}{2}\right) \times \dfrac{32}{44} = 0.18720$ "

$\overline{\qquad\qquad}$
0.34776 "

A quicker solution is to note that CO^2 represents O^2, its own volume of oxygen, and CO represents O, or half its volume of oxygen, and since 1 cubic meter of oxygen weighs $0.09 \times (32 \div 2) = 1.44$ kilograms, the weight of oxygen required is

$$\left(\dfrac{0.223}{2} + 0.13\right) \times 1.44 = 0.34776 \text{ kg.}$$

The total oxygen in 448.5 m^3 of gases is therefore 448.5×0.34776 $= 156$ kg.
subtracting that furnished by the solid charges $= 55.6$ "
there remains oxygen furnished by blast $= 100.4$ "
If the blast were perfectly dry air, its nitrogen would be $100.4 \times (10 \div 3)$ $= 334.7$ "
and the weight of dry blast $= 435.1$ "
and its volume at 0° C. $435.1 \div 1.293$ $= 336.5$ m^3

The blast, however, is not dry, and the 100.4 kilos. of oxygen includes that brought in by the moisture. The moisture weighs 5.66 grains per cubic foot of measured dried air; it is, therefore, simplest to calculate the oxygen entering the furnace per unit volume of air proper entering. Since 5.66 grains $= 5.66 \div 437.5 = 0.01294$ ounces av., the calculation can be made in ounces per cubic foot or kilograms per cubic meter in identical figures as follows:

Oxygen in 0.01294 parts of water $= 0.01294 \times \dfrac{8}{9} = 0.0115$

Oxygen in 1 volume of air proper, at 24° C.

$= 1.293 \times \dfrac{3}{13} \times \dfrac{273}{273 + 24}$ $= 0.2743$

$\overline{\qquad\qquad}$
Sum $= 0.2858$

The blast received, per 100.4 kg. of oxygen thus received, is therefore, measured dry at 24° C:

$$\dfrac{100.4}{0.2858} = 351.8 \text{ cubic meters,}$$

and in cubic feet, per 100.4 pounds of pig iron:

$$\dfrac{100.4 \times 16}{0.2858} = 5628.8 \text{ cubic feet.}$$

The volume of the moist air containing this will be the volume of assumed dried air plus the volume of the water vapor. The latter is, per unit volume of dried air:

$0.01294 \div \left(0.09 \times \dfrac{18}{2} \times \dfrac{273 + 24}{273}\right) = 0.01738$

and the volume of moist air received, at 24°, is, therefore,

$351.8 \times 1.01738 = 357.9$ cubic meters.
Or $5628.8 \times 1.01738 = 5726.6$ cubic feet.

The weights of water, oxygen and nitrogen received per 100 of pig iron are (as already given on the balance sheet):

H^2O $0.01294 \times 351.8 = 4.52$ kg.
O^2 $0.2743 \times 351.8 = 96.4$ "
N^2 $0.9143 \times 351.8 = 321.3$ "

And the volumes of these, considered theoretically at 0° C. and standard pressure, per 100 kg. of pig iron made:

H^2O vapor $4.52 \div 0.81$ $=$ 5.6 cubic meters.
Air $417.7 \div 1.293 = 322.8$ "   "

$$Sum = 328.4 \text{ " "}$$

There are several other modifications of this solution which will suggest themselves to anyone who studies out the question, but while half a dozen ways may be equally correct, that one is logically to be preferred which is most easily understood and is the shortest. One solution, however, based on volume relations, is well to know. Water vapor, H^2O, represents half its volume of contained oxygen, while air has 0.208 oxygen. The cubic foot of dried air at 24° C. was accompanied by

$$0.01294 \div 0.81 \times \frac{273 + 24}{273} = 0.0174 \text{ cubic foot}$$

of moisture, which was removed in making the test. The oxygen per cubic foot of dry blast was, therefore, at 24° C.:

O as $H^2O = 0.0174 \div 2$ $= 0.0087$ cubic foot.
O^2 as air $= 1.0000 \times 0.208 = 0.2080$ "   "

$$0.2167 \text{ " "}$$

Weigh. of this oxygen:

$$1.44 \times \frac{273}{273 + 24} \times 0.2167 = 0.2868$$

And volume of dry blast, at 24°, per 100 of pig iron:

$$\frac{100.4}{0.2868} = 350.1 \text{ cubic meters.}$$

The difference of about 0.4 per cent between this and the previously-calculated result is due to the figures not being carried out to a sufficient number of decimal places.

(4) *The efficiency of the blowing plant* is found by calculating the volume of air and moisture received by the furnace per minute, calculated to 24° C. = 75° F., and comparing this with the piston displacement of 40,000 cubic feet per minute.

Volume of moist air received, at 24° C.,
per 100 lbs. of pig iron made $= 5726.6 \text{ ft.}^2$
Volume per day $= 5726.6 \times 8019.2$ $= 45,922,750 \text{ ft.}^2$
Volume per minute $= 45,922,750 \div 1440 =$ 31,891   "

$$\text{Efficiency of blowing plant } \frac{31,890}{40,000} = 79.7 \text{ per cent (4)}$$

(5) *The heat balance sheet* is based for most of its data upon the balance sheet of materials, the calculations already made and the additional data given in the statement.

The balance sheet shows 80.3 kilos. of carbon oxidized in the furnace. Of this, the amount oxidized to CO and remaining as such in the gases is obtained from the amount of CO in the gases, as follows:

C in CO $= 448.5 \times 0.223 \times 0.54 = 54.0$ kg.
C oxidized to CO^2 is therefore $80.3 - 54 = 26.3$ kg.

The heat in hot blast is calculated from its volume assumed to be at 0° C., and already calculated, viz.: 322.8 cubic meters of air proper and 5.6 cubic meters of water vapor, with mean specific heats per cubic meter between 0° and 382° C. of 0.313 and 0.40 respectively.

The heat of formation of the pig iron may be taken from the amount of carbon in the iron, as 705 Calories per kilogram of carbon, and that of silicon omitted from consideration.

The heat of formation of the slag, since it contains 29.5 of silica and alumina to 28.5 of lime and magnesia, may be taken as 150 Calories per unit of silica and alumina.

The total heat received and generated, and to be accounted for in the furnace, is therefore, per 100 kg. of pig iron:

Carbon to CO 54.0×2430 $= 131,220$ Calories.
Carbon to CO^2 26.3×8100 $= 213,030$ "
In H^2O vapor $5.6 \times 0.40 \times 382$ $= 39,385$ "
In air proper $322.8 \times 0.313 \times 382$

Solution of C in iron 4×705 $=$ 2,820   "
Formation of slag 29.5×150 $=$ 4,425   "

$$\text{Total} = 390,880 \text{ "}$$

The items of heat distribution are 325 Calories in each kilogram of pig iron, 525 Calories in each kilogram of slag, heat in the waste gases at 281°, heat in cooling water, lost by radiation and conduction (by difference), evaporation of the moisture in charges, expulsion of CO^2 from carbonates, decomposition of moisture of blast, reduction of silicon and iron.

Reduction of Fe   95 kg. $\times 1,746$ $= 165,870$ Calories.
Reduction of Si   1 " $\times 7,000$ $= 7,000$ "
Expulsion of CO^2 from
    $CaCO^3$ 16.7 " $\times 1,026$ $\Big\}$
Expulsion of CO^3 from $= 18,666$ "
    $MgCO^3$ 2.3 " $\times 666$
Evaporation of H^2O 18.7 " $\times 606.5$ $= 11,342$ "
Heat in waste gases:
    N and CO   400 m³ $\times 0.3106$
    CO^3 58.3 m³ $\times 0.446$ $\Big\} \times 281° = 43,836$ "
    H^2O 23.1 m³ $\times 0.382$
Decomposition of moisture of blast:
    H^2O 4.5 kg. $\times (29,040 \div 9)$ $= 14,511$ "
Heat in slag 58 kg. $\times 525$ $= 30,450$ "
Heat in pig iron 100 $\times 325$ $= 32,500$ "
Heat in cooling water (300,000 gallons
    per diem) 300,000 $\times 8.3$ (lbs.) $\times$
    50° $\div 8019.2$ $= 15,525$ "
Loss by radiation and conduction (difference) $= 51,180$ "

$$\text{Total} = 390,880 \text{ " (5)}$$

(6) *The proportion of the fixed carbon burned at the tuyeres* is obtained by calculating the weight of carbon which the oxygen entering at the tuyeres could oxidize to CO, as follows:

Oxygen entering at the tuyeres $= 100.4$ kg.
Carbon burned to CO $= 100.4 \times \dfrac{12}{16}$ $= 75.3$ "
Fixed carbon charged $= 84.3$ "
Proportion burned at tuyeres $= 89.3$ per cent.

A more just proportion is that between the carbon burned at the tuyeres and the total fixed carbon oxidized, because the fixed carbon which carbonizes the iron cannot be oxidized under any circumstances. This proportion in this furnace is:

$$\frac{75.3}{84.3 - 4.0} = 93.8 \text{ per cent.} \tag{6}$$

Indicating a very fair approximation to Grüner's ideal working. If we make the further allowance, that the silicon in the pig iron is necesarily reduced by solid carbon, and that therefore the solid carbon needed to reduce the 1 kilogram of silicon $(1 \times (24 \div 28))$ is in no case available for combustion at the tuyeres, we have the approach to Grüner's ideal working measured by the ratio

$$\frac{75.3}{80.3 - 0.9} = 94.8 \text{ per cent,}$$

in spite of which, however, the furnace is not doing very good work.

(7) The proportion of the heat requirement generated or available at or about the tuyeres is determined as follows:

Combustion of C to CO $= 75.3 \times 2430 = 182,979$ Calories.
Heat in hot blast $= 39,385$ "
Formation of pig iron $= 2,820$ "
Formation of slag $= 4,425$ "

$$\text{Total} = 229,609 \text{ "}$$

Against which must be charged heat re-
quired to decompose moisture of blast $= 14,511$ "

Leaving net heat available $= 215,098$ "
which is $215,098 \div 390,880 = $ **55** per cent.

of the total heat generated and available in the furnace.

Another way in which it is sometimes put, is that the oxidation of carbon to CO or CO^2 furnishes a certain amount of heat to the furnace (346,250 Calories in this case), of which a certain amount is generated by combustion at the tuyeres (182,979 Calories), making the ratio thus considered, in this case, 53 per cent,—which is almost the same figures as above, but not so significant, since it is illogical to consider the heat brought in by the hot blast as not being available heat for doing work in the tuyere region.

(8) The proportion of iron reduced by solid carbon is found by finding how much carbon is used up in that reduction.

Fixed carbon charged $= 84.3$ kg.
Fixed carbon carbonizing the pig iron $= 4.0$ "

Fixed carbon oxidized in the furnace $= 80.3$ "
Fixed carbon oxidized by the blast $= 75.3$ "

Fixed carbon oxidized above the tuyeres $= 5.0$ "
Carbon needed to reduce 1 kg. silicon $= 0.9$

Carbon used for reducing FeO $= 4.1$ "

Amount of Fe. thus reduced $= 4.1 \times \dfrac{56}{12} = 19.1$ "

Proportion of the total Fe. thus re-
duced $= \dfrac{19.1}{95} = $ **20** per cent. (8)

(9) The maximum temperature of the gases in the region of the tuyeres is that temperature to which the heat there available will raise the products of combustion. This question is best resolved by simply considering the combustion of 1 kilogram of carbon, evolving 2430 calories, while the heat in the hot carbon itself just before it is burnt, and that in the hot blast required, will also exist as sensible heat in the products,—the whole diminished by the heat necessary to decompose the water vapor blown in.

Since, per 75.3 kg. of carbon burned at the tuyeres there enter 5.6 m^3 of water vapor and 322.8 m^3 of air proper, measured at $0°$, the volume of blast per kilogram of carbon oxidized at the tuyeres is

H^2O $5.6 \div 75.3 = 0.0738$ $m^3 = 0.0598$ kg.
Air $322.8 \div 75.3 = 4.2869$ m^3

The products of the combustion are, per kg. of carbon burned:

CO $22.22 \div 12$ $= 1.8519$ cubic meters.
N^2 $321.3 \div 1.26 \div 75.3 = 3.3865$ "   "
H^2 equal to H^2O decomposed $= 0.0738$ "   "

Total $= 5.3122$ "   "

The heat available to raise their temperature is:

Heat of combustion of 1 kg. carbon $= 2430$ Calories.
Heat in hot blast $= 39,385 \div 75.3 = 523$ "
Heat in hot carbon at $t°$ (or very nearly) $= 0.5t - 120$ "
Less heat absorbed in decomposing steam

$14,511 \div 75.3 = 193$ "
Net heat available in gaseous products $= 2640 + 0.5t$ Cal.
Calorific capacity of gaseous products
$= 5.3122 (0.303t + 0.000027t^2)$
Therefore $5.3122 (0.303t + 0.000027t^2) = 2640 + 0.5t$
Whence $t = 1910°$ (9)

This represents the absolute maximum of temperature which the gaseous products formed at the tuyeres can possess.

(10) If all the moisture were removed from the blast, the heat available would be::

By combustion of 1 kg. carbon $= 2430$ Calories.
Heat in 4.4685 m^3 of dry air at $382°$ C.
$= 4.4685 \times 0.313 \times 382 = 574$ "
Heat in 1 kg. of carbon at $t°$ $= 0.5t - 120$ "
Net heat available in gaseous products $= 0.5t + 2884$ Cal.
Calorific capacity of gaseous products
$= 5.3976 (0.303t + 0.000027t^2)$
Therefore $5.3976 (0.303t + 0.000027t^2) = 2884 + 0.5t$
Whence $t = 2018°$ (10)

It is to be noted that this is $108°$ C. $= 194°$ F. higher than with moist blast; and while the slag and iron in passing through this zone of high temperature will not reach this maximum temperature, yet they would be heated approximately $100°$ C. higher when using dry blast, if all other conditions were kept constant.

N. B.: In working this problem, the metric measurements and English measures have been purposely used interchangeably, in order that readers may understand better that if weights are taken in pounds and the heat unit is the pound Calorie ($1°$ C.), the same numbers represent a solution in either system. When volumes are concerned, cubic feet and ounces, or ounce calories, have practically the same numerical expression as cubic meters and kilograms or kilogram calories.

Gold and Silver Refining.

The electrolytic refining of gold has been carried out on a large scale for a number of years in Germany, and was first applied in this country in the United States Mint in Philadelphia by their smelter and refiner, Dr. D. K. Tuttle. An authoritative illustrated description of the equipment of the electrolytic refining room of the Philadelphia Mint, from the pen of Dr. Tuttle, may be found on page 157 of our Vol. I., while a more detailed discussion of the electrolytic process of gold refining was given by the original inventor of the process, Dr. E. Wohlwill, of Germany, on pages 221 and 261 of our Vol. II.

The annual report of 1905 of the director of the United States Mint contains some interesting notes by Dr. Tuttle on the progress made at the Philadelphia Mint in electrolytic refining, which are herewith reproduced verbatim:

"Operations in the nitric acid refining plant were discontinued in February, 1905. The electrolytic process for refining gold continues to give satisfaction, and now that the method for parting and refining silver is perfected, all the refinery operations are conducted by electrolysis.

"At the date of the last annual report electrolytic refining was confined to high-grade gold containing a small percentage of silver. Since that time the hope has been realized which was expressed in the report for 1902 (p. 123), where it was said: 'The ideal refining plant for a mint would be one in which electrolytic separations are the leading features. The bullion to be parted and refined will be divided into two classes, by selection and by blending in making up the materials for anodes. The one class will have silver as the predominant metal, but carrying as much gold as will permit its treatment by electrolysis in a silver bath. The product from this operation will be fine silver, and the residues will consist of gold, platinum and other impurities. The other class of bullion is to be largely gold, as at present, selected for electrolytic treatment in a gold chloride solution. The residues from each process would pass to the other for final treatment. Work is now being done looking to a realization of this scheme.'

"By persistent effort this has now been accomplished, and the energy obtained from a few bushels of coal in the form of an electric current is made to do the work of dray loads of expensive acids.

"Doré bars of silver containing small quantities of gold are successfully refined in industrial establishments by the Moebius or similar processes, but, since in mint practice silver has to be added to the gold and used as parting material, an economical process must require the minimum percentage of silver in the anodes. As will be seen from the following description, our anodes in the silver cells consist of 30 per cent gold and 70 per cent of silver, copper, lead, etc. It is believed that the successful treatment of such bullion on a large scale by electrolysis is an innovation in practice.

"In the electrolytic plant installed in the Philadelphia Mint two distinct processes are employed, the choice of one or the other depending on the character of the bullion to be treated. If it be gold, high in fineness, but containing from 40 to 60 parts of impurity per thousand, such as silver, platinum, copper, lead, etc., it is refined by what is known as the 'Wohlwill process." * * * *

"When silver is the chief element, with lesser percentages of gold, copper, lead, etc., present to be parted, a different electrolyte and a modified system of working are employed. The electrolyte used is a 3 per cent solution of silver nitrate in water, to which is added $1\frac{1}{2}$ per cent free nitric acid. The tanks are of earthenware, 40 inches by 20 inches and 11 inches deep. In each of these are suspended from conducting rods 42 anodes and 40 cathodes. The anodes are composed of 300 parts of gold in 1,000, the remaining 700 parts consisting of silver, copper and other impurities as parting material. They are cast into bars $7\frac{1}{4}$ inches long by $2\frac{1}{2}$ inches wide and three-eighths of an inch thick. The cathodes are strips of fine silver of same length and width rolled to 0.016-inch thick.

"Eight cells so equipped are connected up in series, and a current with a density of 0.05 amp. per square inch passes through the system. The silver and other soluble metals are extracted from the anode by the combined action of the current and electrolyte, while the gold remains as a chocolate-brown substance, sufficiently coherent to retain the original form of the anode. Meanwhile pure silver is deposited in a crystalline but coherent form on the cathode. Heretofore a coherent deposit has not been obtained from a silver nitrate solution, the product in the Moebius and other processes in commercial use being non-adherent crystalline granules, which fall from the cathode to the bottom of the cell.
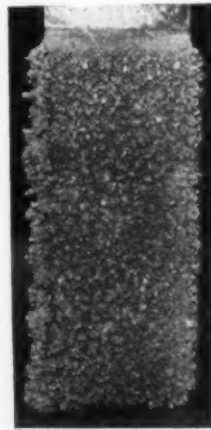
ELECTROLYTIC SILVER CATHODE.

"The deposit in a coherent form is due to a happy observation of the melter and refiner, in which it was discovered that the addition of a very small amount of a colloid, such as gelatine, to the electrolyte changed completely the nature of the deposit, so that the "vertical system" of anodes and cathodes became for the first time possible.[2] The cathodes are washed with water, melted without fluxes and cast into bars.

"The anodes retained persistently a small amount of silver, even if subjected to the current after oxygen is freely evolved from their surfaces. A fact new to the scientific world should here be noted, namely, that if the action of the current be pro-

[1] The description which now follows is essentially the same as in Dr. Tuttle's article, our Vol. I., page 157.

[2] See also the paper by F. D. Easterbrooks, on parting of bullion, in our Vol. III., page 373. According to a private communication from Dr. Tuttle, his silver cathodes are often 1000 fine. One of the same is illustrated in the above diagram. Concerning the anodic information of gold teroxide, mentioned in the next paragraph, Dr. Tuttle remarks that this is, of course, analagous to the formation of lead and silver peroxides under similar conditions.—Editor.

longed on the anodes, after most of the silver has been dissolved, the nascent oxygen evolved will attack the spongy gold and produce a small but notable quantity of gold teroxide, soluble in concentrated nitric and sulphuric acids. It is deposited from these solutions on dilution, but, of course, in a finely divided form. The liability to its occurrence should be known to the operator. It is probably a hydrated oxide, since by simply heating the oxidized anode to 250° no gold passes into solution in acids.

"The silver remaining in the anode is removed by a treatment in hot nitric acid, the resulting solution being used to replenish the electrolyte. The gold is then thoroughly washed with water and melted.

"If platinum be present it will remain with the gold, and we usually pass this through the gold-refining cells, which is a very inexpensive operation, and gives a much purer product while affording the means for recovering other values. The losses in these operations need be very slight if care and cleanliness be used. All accidental sloppages of solutions are mopped with cotton cloths, which are burned and the ashes preserved. The necessary losses should be less than 1 ounce in 10,000. How much less we hope to show by precise figures after more extended experience."

The Lines of Current in Storage Batteries—An Experimental Study.

BY M. U. SCHOOP.

Small plates of spongy lead are very suitable as testing electrodes for the separate determination of the capacities of the positives and negatives of a lead accumulator. In this case the little testing plate is generally placed in the electrolyte above the upper rim of the plate, proper connections being made to a galvanometer. The chemical reaction consists in the discharge of the little plate of spongy lead by means of the current passing through the instrument, whereby the lead is slowly changed into sulphate.

What happens if the same little testing plate is placed between the main electrodes right into the field of the lines of current, no connections being made to a galvanometer? Like any intermediate electrode the spongy lead plate will act as a bipolar electrode with anodic polarization on one side and cathodic polarization on the other side. But since we have here something like a short-circuited couple the products of polarization will tend to destroy each other immediately, and the final result of the electrochemical reaction on the plate is a slow formation of sulphate or, what is the same, a slow discharge. It may be easily seen that the speed of this reaction depends on the thickness of the plate, that is, on the voltage drop within the plate.

In order to investigate a storage battery electrode showing a non-homogeneous discharge, it would therefore suffice to place two identical small spongy lead plates into the electrolyte on both sides of the electrode in exactly the same distance from its surfaces, and to observe the reading of a sensible galvanometer connected to the two testing plates. If the galvanometer needle shows a deflection, one of the two little testing plates is met by a greater number of lines of current than the other one.

We may, for instance, combine two parallel small spongy lead plates to a little instrument, as shown in Fig. 7, which explains itself. It is, of course, highly important that the distances of the two little testing plates from the main electrode should be exactly the same. The best arrangement is to let the two little testing plates rest directly on the support, as shown in Fig. 8, a direct contact of the little testing plates with the main electrode to be tested being avoided by small perforated hard rubber pieces pasted on the rim.

Such a measurement will establish whether the current density is the same on both sides of the electrode. As may be stated at once, this is never the case in those traction cells with n positive and n + 1 negative plates, in which the negative end plates do not differ from the other negative plates with respect to thickness and capacity.

On that side of the positive end plates, which looks towards the negative end plates, the current density is considerably greater than on the opposite side, since it would be wrong to suppose that the terminal negatives are active on their inner side only.

With large stationary types of storage batteries, this fact has been taken into account long ago, at least in good products, although perhaps primarily for the sake of saving lead. With transportable cells, however, terminal negatives with half capacity are not used in practice to my knowledge. Since the electrodes of portable cells, especially of traction cells, must be very thin, anyhow, for the sake of a high capacity, per unit of weight, it is quite difficult to select an arrangement by which a uniform discharge of all the plates can be attained. It might, perhaps, be possible to design the connections of the end negatives with the cross-bars, so as to increase their resistance.

FIG. 7.
ANALYZER.

As may be expected, and as is often found in practice, the end positives of discarded traction cells show far more signs of disintegration than the other positives, and often the end positives are more or less curved toward the terminal negatives in the manner indicated in Fig. 9.

In traction cells the current is introduced generally in the middle of the cross-bar which connects the different plates. This results evidently in a drop of voltage whereby the end plates carry less current than the plates in the center of the cell. The double capacity of the negative terminal plates, however, more than counterbalance this effect of the voltage drop.

With the little instrument described above it is easy to prove in an easy manner that this disturbing effect of the terminal negatives relates not only to the positives next to them but to all other plates. Of course, the readings of the millivoltmeter always get smaller the nearer we come to the center of the cell, and they disappear altogether at the center where the current is introduced into or taken off from the cell.
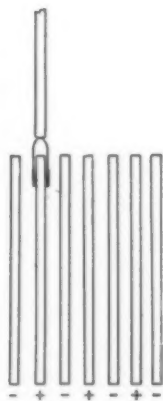
FIG. 8.—USE OF
ANALYZER.

Concentration differences annihilate themselves, first, because the drop of concentration in the pores of the plates causes diffusion from higher to lower concentration, and, secondly, because so-called concentration currents are produced which are even more effective in bringing about uniformity of concentration. Dolezalek has shown this mathematically while Schoop has proven it by experiment.[1]

At this place we may mention another kind of local currents which always occur when the charge or discharge of a plate is not homogeneous (i. e., in all cases of practice). The part of the plate which is more strongly discharged (the bottom part) contains more dilute acid within the pores while the less discharged parts of the plate (the upper parts) have a stronger concentration of acid in their pores. By diffusion and concentration currents these differences of concentration tend to annihilate themselves until equilibrium is reached. It is easy

to prove this experimentally by the little instrument with the two little spongy lead plates described above. But the needle of the galvanometer gives larger deflections if the little testing plates are not placed behind each other, but at a distance of about 8 mm. side by side. What is now measured is simply the voltage drop corresponding to the distance of the little plates in the electrolyte. If we now turn the instrument slowly around its longitudinal axle until the galvanometer shows zero current, we have reached an equipotential curve, on which the lines of current must be perpendicular.

The current density is, of course, a maximum in next proximity to the surface of the main electrode and decreases gradually with increasing distance from the main electrode. The figure of the lines of current from a spongy lead electrode which is not homogeneously discharged at the top and bottom. looks about like that shown in Fig. 10. For the investigation of these currents it is easier to place the electrode which is being tested longitudinally on the side. If it is left in its normal perpendicular position the analyzer is to be modified to suit the purpose.
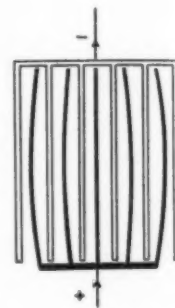

FIG. 9.—DISTORTION OF
BATTERY PLATES.

The presence of such very strong parasitic currents can be proven, if, for instance, a charged spongy lead plate is exposed to about one-third of its surface for a few minutes to the oxidizing action of the air. But the same phenomenon may also be observed whenever there is any lack of homogeneity at different places of an electrode in a cell.

This simple method gives us reliable means of testing whether the charge or discharge of an electrode is homogeneous. I have used the same experimental method for proving that any electrolytic shunt results in a lack of homogeneity of the distribution of current density over the surface of an electrode, and that, for instance, in a storage battery, on account of the space left between the lower ends of the plates and the bottom of the cell, the current density is higher in the lower parts of the plates than in the upper ones. In storage battery factories this point is of importance for the formation of plates.

As has already been mentioned the straying of lines of current is only one of several factors which produce a non-uniform distribution of the current density over the electrodes. The effect of the straying of the lines of current may be more or less masked or counterbalanced by other factors, like the construction of the lead grid, or the mechanical details of the

FIG. 10.—DIAGRAM OF EQUIPOTENTIAL
CURVES.

connections between the plates and the external circuit, or the porosity of the active masses.
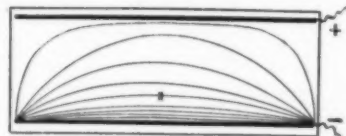
In general, an electrode will tend to correct itself any lack of uniformity of current density over its surface. For instance, during discharge, those parts of the electrode which have a higher current density will get a greater resistance, so that the current will gradually pass over to those parts which were less discharged before. On the other hand, during charge, those places which have first been charged will have more concentrated acid in the pores, and this will result in a counter e. m. f. which will force the lines of current out of their position.

It would, therefore, be wrong to think that the distribution of the lines of current over an electrode or at a certain place of the electrode surface or of the cross-section of the electro-

[1] "Electrochemical Industry," 1904, Vol. II., pages 272 and 310.

lyte remains always the same. On the contrary, there will be continuous and gradual changes in the current density at any place, and I may say right here that they are far more evident during a discharge than during a charge.

In order to prove the fluctuations of the current distribution in a working storage battery and to get exact figures, we may again use the little instrument described above, which is placed at equal time intervals between the plates (at the top, in the middle or at the bottom). With large cells which have a considerable distance between plates this instrument may be used without modification.

With traction cells with thin plates, however, the distance between the plates is mostly 3 or 4 mm. only, and is too small to employ the little instrument directly. It is, therefore, neces-

After 7¼ hours the discharge was interrupted for the night, and was continued the next morning. The changes of concentration, due to diffusion during the night, are quite remarkable. The plate-interval which had the lowest current density the evening before has now the highest. The different plate-intervals have just reversed their places.

This series of tests also shows clearly that the end positives have a greater current density than the other positives (see the curve of the first plate-interval).

The curves at the top of the diagram, Fig. 11a, relate to different plate-intervals, all being tested at the top. In Fig. 11b (the bottom of the diagram) the differences of current density in the same plate-interval at the top and at the bottom are plotted graphically. It will be seen how the difference of
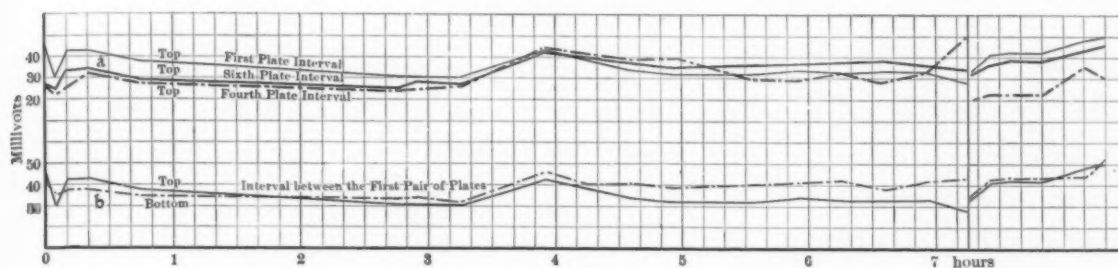
FIG. 11.—CURVES SHOWING CHANGES IN DIFFERENT INTERVALS BETWEEN PLATES.

sary to place the plates at a greater distance from each other, using glass tubes or hard rubber forks for keeping the distance constant. The glass tube of the analyzer is provided with marks which show the depth to which the little testing plates are immersed into the electrolyte.

I may say that the polarization produced at the little testing plates is very small, and amounts to only 1 or 2 millivolts. To measure it, one has to open the circuit after the measurement or, what is preferable, to place the analyzer into a nearby vessel containing acid of the same concentration, and to observe simultaneously the millivoltmeter.

The curves shown in Fig. 11 relate to a traction cell with five positive and six negative plates, 300 x 100 x 3 mm., with a distance of 15 mm. of the plates from the bottom. This cell as tested, differs from a normal traction cell only in so far as the plates are placed at a distance of 14 mm. instead of 3 mm., the uniform distance between the plates·being produced by celluloid combs instead of corrugated, perforated hard rubber insulators. This cell had, therefore, in the whole ten intervals between the plates; each interval was to be tested at three different places, at the top, in the center and at the bottom, with respect to current density. This meant thirty measurements. It is preferable to have two observers, one for the millivoltmeter and the other for the analyzer. In order to get reliable results which agree with each other, the observers should have some experience in working together.

The results may, of course, be plotted graphically in different ways.

The Fig. 11a (top of the diagram) shows, for instance, the changes of current density during an 8-hour discharge in three different plate-intervals, all the measurements being made at the top, that is, slightly below the surface of the electrolyte. One sees at a glance that the distribution of the current is subjected to very great changes, the curve of these changes having a decided wave form.

Observations of the fourth plate intervals show this very clearly. The current density is here a minimum at the beginning, but the curve rises rapidly after 2½ hours and goes down again after another hour and a half, and only near the end of discharge it goes up again rapidly.

Similar, but in the opposite direction, are the changes of the first plate-interval.

the readings gets greater and greater toward the end of discharge (15 millivolts). But in this case the night's rest also renders the concentrations more uniform at the top and the bottom, so that the next morning the two plates continue to discharge with about the same current densities at the top and the bottom.

IV.

The described current fluctuations of a pair of plates occur always. This may be proven by still another arrangement. Between the electrodes to be tested are placed in exactly symmetrical distance from the same two little testing plates connected to a millivoltmeter. The current is closed, and the position of the testing plates is so adjusted that the instrument shows zero current at the beginning, whereby it is of no account what the relative distance is between the little testing plates.

The arrangement is somewhat similar to a Wheatstone bridge, since the bridge will carry no current as long as there is no change in the branches of the circuit during the experiment.

In a great number of experiments under the most different conditions (new and old electrodes, large and small distance between the plates, with or without electrolytic shunt), it could be established that always and under all circumstances, the conditions of the current will change, and that these changes are in general very much more evident with spongy lead plates than with lead peroxide plates.

This may be due to the well-known fact that spongy lead is subjected to changes of structure, and that the contact with the support has a tendency to get poorer and poorer with each charge or discharge, while with the positive plates the PbO_2 layer which grows into the lead guarantees an excellent contact. Moreover, according to the investigation of Streintz PbO_2 has a conductivity about half that of mercury, and is, therefore, to be considered as an excellent conductor.

If we want to get information concerning the behavior of a single plate we may use a method similar to that in determining the capacity of a single positive or negative plate. This method consists essentially in using as opposite electrode a plate of four or five times greater capacity, so that the changes of current density occuring on the opposite electrode are neg-

ligible compared with those on the other plate of smaller capacity.

As an example I give the tests of a negative lead electrode, the formation of which had been started according to Planté. It was alternately charged and discharged against a normal positive electrode in ordinary acid. The opposite side of the negative Planté plate was covered with a layer of insulating
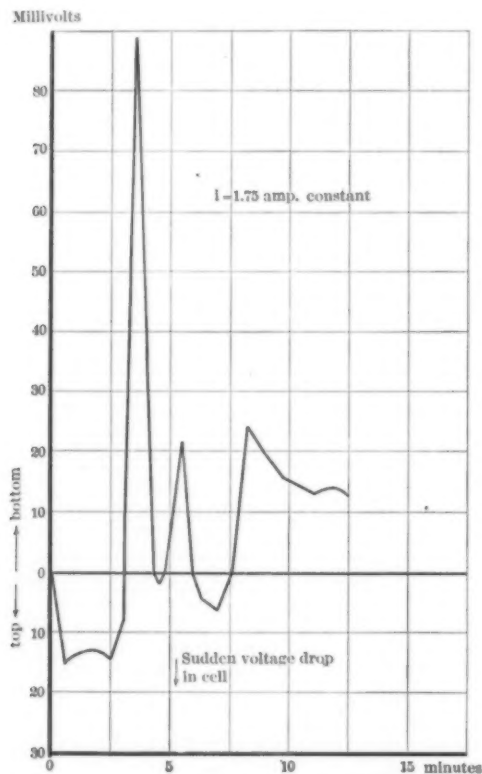

FIG. 12.—DIFFERENCES BETWEEN THE BEHAVIOR OF PLATE
AT TOP AND BOTTOM.

paint and the plates were otherwise so arranged as shown in Figs. 1 and 2 (page 269, July issue).

It is hardly necessary to point out that the picture of the distribution of the lines of current is immediately changed if anything whatever is changed in the conditions of the experiment—whether it be the connections to the plates through which the current is conducted to the battery, or the distance between the plates or the space available for the electrolytic shunt, etc. A change would also be found immediately when the Planté spongy-lead plate would be replaced by a positively-formed lead plate.

As Fig. 12 shows, the upper parts of the plate where the current enters the plates, carried a higher current density in the beginning. After 4 minutes, however, a very sharp maximum in the opposite sense appeared, which was almost coincident with a sudden drop of voltage of the cell.

On account of the relatively high current strength of 1.75 amps. the changes in the distribution of the current over the Planté plate were very marked.

A control of the measurements of the different current densities at the top and the bottom of the plate may also be obtained by simply observing the plate, especially the changes of color of the active masses and the beginning of gasing.

As may be seen from what has been said, the "current-difference curve" is the resultant of two component currents,

of which the one current is the real difference current, while the other is to be considered as a local current between the different places of the non-uniformly discharged Planté plate, which is superposed in positive or negative sense to the other current.

In order to find the value of the latter and to separate it from the current difference proper, we could, for instance, open the circuit after each observation and could make a second measurement immediately after the first. For the elimination of the parasitic currents, however, it appeared to me more suitable to replace the bridge method by an arrangement in which the single little plates are replaced each by pairs of little plates. In this case instead of a one-measurement two must be made immediately after one another. What is measured is again the voltage drop in the electrolyte, which is proportional to the current density at that place.

An experiment of this kind with normal and very carefully prepared storage-battery plates for traction cells will finally be described.

As will be seen from the small sketches in Figs. 13 and 14, showing the arrangement of the experiments, the second experiment (Fig. 14) differs from the first experiment (Fig. 13) only in the fact that in Fig. 13 the arrangement is absolutely symmetrical, while in Fig. 14 an electrolytic shunt is provided only on one side (b). In both cases the position of the two analyzers is exactly the same, so that any difference in the readings of the millivoltmeter must be due entirely to the differences in arrangement of the electrolytic shunt.

In both figures the curve a^1 corresponds to the readings made with the analyzer at a, while curve b^1 corresponds to the read-
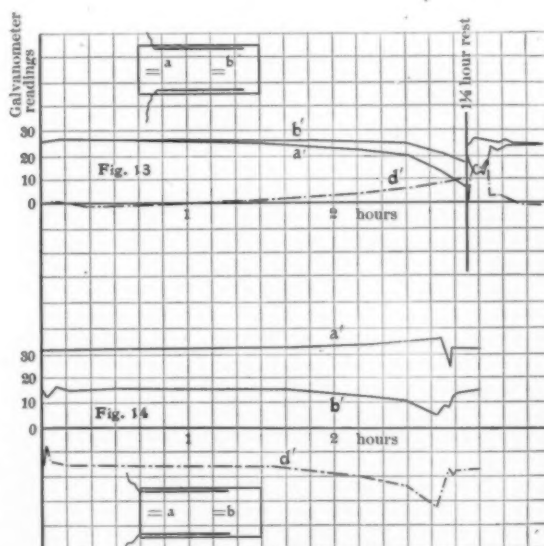
FIGS. 13 AND 14.—COMPARATIVE EXPERIMENTS SHOWING THE
EFFECT OF AN ELECTROLYTIC SHUNT.

ings with the analyzer at b, while the dotted line d^1 represents the difference between the current densities ($b^1 - a^1$).

To avoid any misunderstanding, I may emphasize that these curves are not intended to give any information on the current densities on the electrode surfaces, but that they demonstrate the conditions in the electrolyte at the places a and b. They show how the distribution of the current in the electrolyte is subject to changes as soon as the smallest change is made in the arrangement of the experiments. The following figures, which give the results of both tests, will show the method of the measurements:

FIRST TEST.
READINGS OF MILLIVOLTMETER.

Voltage at Terminals.	Time. Hrs. Min.	d'	a'	b'	i amps.
1.69	0	0	25½	25½	8.0 constant
1.67	2	½	26	26½	
...	7	0	26¾	26¾	
1.66	13	0	26½	26½	
1.41	21	½	26¼	26	
1.45	32	¼	26 1/5	26	
1.4	55	0	25½	25½	
...	1.35	1½	24	25½	
1.34	2.07	3¼	21¾	25	
1.32	2.16	4	21	25	
1.28	2.30	5½	19	24½	
1.3	2.46	8	11½	19½	

1½ HOURS REST.

...	0	0	23	23	
1.4	1	7½	16	23½	
1.2	2	13	13	26	
1.0	5	14½	12	26½	
Polarization rev'd	8	16½	9	25½	
1.2	8	2	23	25	
1.15	12	3	21½	24½	
...	15	2	23¼	25½	
...	20	½	25	24½	

SECOND TEST.

Voltage at Terminals.	Time. Hrs. Min.	d'	a'	b'	i amps.
...	0	16	31	15	8.0 constant
...	1	8 2/3	31 1/3	12 1/3	
1.73	3	14½	31	16½	
...	5	15	31½	16½	
1.67	27	15¾	31¾	16	
1.6	1.38	17¼	32½	15¼	
1.53	2.15	21	33½	12½	
1.49	2.30	24	34½	10½	
1.13	2.45	24¼	33½	9¼	
0.5	2.47	17	25	8	
Polarization rev.	2.48	19	32	13	
1.03	2.50	18	32	14	
0.93	3.00	17½	32	15½	

To sum up, the chief results of this investigation are as follows:

1. The ions which transport the electricity through an electrolyte do not select the "geometrically shortest" path, but the "electrically shortest" path, so that Ohm's law for metallic wires of uniform cross-section can be applied to electrolytes only in the special case, if there is no straying or distortion of lines of current.

2. In all those cases (that is, in all cases of practice) where the cross-section of the electrolyte is larger than the surface of one electrode, there is an "electrolytic shunt"; the lines of current no longer pass in a straight direction through the electrolyte, and the distribution of the current density over the electrode surfaces is no longer uniform. What I call the "electrolytic shunt" really acts as a shunt, i. e., has the effect of diminishing the resistance, as may be shown by the use of a sensible voltmeter.

3. A large number of the troubles experienced with storage battery plates, like distortion of the plates, dropping out of active material, may be easily explained in accordance with what was said above under 1 and 2, since on account of the layer of acid between the lower rims of the plates and the bottom of the vessel (which layer of acid acts as electrolytic shunt), the plates carry more current near the bottom than at the top. This non-uniform current density at different parts of the plate results in non-uniform changes of volume. This necessarily results in mechanical distortions. whether we have to do with pasted plates or Planté plates. This is a point

which, up to the present time, has not been properly considered, especially in the formation process.

4. At a storage-battery electrode which is non-homogeneously charged or discharged strong local currents occur. Their existence and direction may be easily proven experimentally by means of suitable analyzers. It seems that lead peroxide plates behave in general far more uniform than spongy lead plates during charge and discharge.

Finally, I will say, that in the present paper I do not claim to have exhausted even approximately the subject. I rather hope that this paper will inspire others to continue these researches. It would be especially desirable if storage battery engineers would pay a little more attention to the important problem of the distribution of the lines of current in storage batteries.

Quarternary Steels.

The Iron and Steel Institute issued at its May meeting (in accordance with its usual practice) the abstracts of the Reports on Research Work carried out by the holders of the "Andrew Carnegie Research Scholarships."

The first of these is by Dr. GUILLET, and deals with quarternary steels. The length even of this official abstract of a report is such that further condensation is necessary in order to indicate the author's conclusions within the limits which can be allotted thereto in the pages of a technical journal.

Nickel-Manganese Steels.—The author divides these into pearlitic, martensitic and γ-iron steels. All attempts of rolling the last-named have completely failed. The transformations which these steels undergo are similar to those already recorded in connection with nickel-steels and manganese steels.

Nickel-Chromium Steels.—The different structures which nickel-chromium steels can present are:

Pearlite, with ferrite or carbide.
Martensite.
Martensite with carbide, and
γ-iron with iron.

The mode of formation of the different structures on the addition of chromium to a nickel steel appear to be as follows:

A. If chromium be added to a pearlitic nickel steel the steel may. to begin with, preserve its pearlitic structure if the amount of chromium added be sufficiently small, and if the sum of the elements Cr + Ni be not very high. Under these conditions the only influence which the chromium exerts is to give the ferrite a much finer grain structure. If the addition of the chromium is sufficiently high, and if the percentages of carbon and of nickel are similarly sufficiently high the structure becomes martensitic.

B. If the chromium be added to a martensitic nickel steel the steel may at first remain martensitic, provided the addition of chromium is sufficiently small. Subsequently carbide is formed, the quantity appearing to depend upon the percentage of carbon, and finally a mixture of martensite, carbide and γ-iron is usually obtained. It sometimes happens that when the percentage of chromium is sufficiently high a structure composed of γ-iron and of carbide is obtained.

C. The addition of chromium to a nickel steel containing γ-iron at first causes no change in the structure, but when the percentage of chromium becomes sufficiently high carbide is formed. This phenomenon commences with a percentage of chromium which is lower in proportion as the amount of carbon is high, and, other things equal, the quantity of free carbide in a steel is higher in proportion as the percentage of carbon and of chromium in the steel rises. In a word, the effect of the chromium is superadded to that of the nickel, with the production of martensite or of γ-iron, but if the percentage of chromium is sufficiently high, carbide is produced in amounts which increase according as the percentages of carbon and of chromium increase. The mechanical properties

of these steels may be deduced from their microstructure, for instance, pearlitic nickel-chromium steels possess a degree of tensile strength and elastic limit which becomes higher the larger the percentage of carbon, nickel and chromium present. The elongations are rather lower than in ordinary steels corresponding to the same percentage of carbon. Martensitic nickel-chromium steels possess exceedingly high tensile strength and elastic limit; the elongations are somewhat low. Nickel-chromium steels containing martensite, together with the double carbide, have practically the same mechanical properties as the martensitic steels. The tensile strength and elastic limit diminish, however, in proportion as the percentage of carbon increases. Nickel-chromium steels with γ-iron have a higher tensile strength and elastic limit than corresponding steels not containing chromium. Their elongation and resistance to shock are somewhat lower and their hardness greater. Steels containing carbide together with γ-iron have practically the same properties as steels containing Y-iron alone, except that their .elongation is rather lower. In short, the properties of nickel-chromium steels resemble those of nickel steels of the same microstructure, except that the addition of chromium somewhat raises the tensile strength and the elastic limit, without, however, much diminishing the elongation and the resistence to shock. In addition to this the hypo-eutectoid chromium-nickel steels containing carbide—a structure which is not met with under similar conditions among nickel steels—are somewhat brittle, while, at the same time, possessing average elongation.

The effects of quenching annealing and case hardening are discussed.

Nickel-Tungsten Steels.—In the case of two series of steels, in which the percentages of carbon and of nickel was such that, had it not been for the tungsten present, they would have been intermediate between the pearlitic and martensite steels, it was found that: (1) The tungsten commences by becoming dissolved in the iron, and favoring, to a slight extent, the production of martensite; (2) when the iron-nickel solution is saturated with tungsten the latter forms a carbide, the quantity increasing with the percentages of tungsten and of carbon present; (3) the addition of tungsten distinctly raises the tensile strength, and slightly raises the elastic limit in normal steels. The elongation, contraction and resistance to shock are lower than in the case of steels not containing tungsten, but vary slightly with the proportion of that element present, whereas, the hardness increases in proportion to the amount of tungsten.

It was also noted that tungsten has a material effect on the mechanical properties of the steel which, after suddenly quenching in water at $870°$ C., possesses an exceedingly high tensile strength and elastic, with a medium elongation, contraction and resistance to shock.

Nickel-Molybdenum Steels.—Concerning these, the author states that: (1) The molybdenum dissolves first of all in the iron and furthers the formation of martensite. In this respect its influence is decidedly greater than that of tungsten. As a matter of fact, no steels containing pearlite were found, whereas, martensitic steels were encountered when the percentage of molybdenum exceeded 2 per cent, a phenomenon not met with in the case of nickel-tungsten steels; (2) when the amount of tungsten present becomes sufficiently high, carbide is formed. In the series of steels under investigation the appearance of the carbide in steels containing the same proportions of nickel and of carbon as the tungsten-nickel series, appeared to take place with the same percentages of molybdenum as it did in the case of tungsten; (3) the influence of molybdenum is of the same nature as that of tungsten, but it acts with very much greater intensity; (4) this difference between the action of tungsten as compared with that of molybdenum is much less pronounced in the case of quenched steels. Commercially, the author does not regard these as interesting or as economically valuable as the nickel-tungsten steels.

Nickel-Vanadium Steels.—The chief point raised is that on quenching a 6 per cent nickel steel, in order to obtain the maximum tensile strength, a very much smaller quantity of vanadium is required than in the case of a steel containing only — 2 per cent of nickel. There is a decided advantage in raising the proportion of nickel as high as possible if it be desired to obtain the maximum mechanical properties after quenching. At the same time it is necessary to preserve the pearlitic structure sufficiently to secure that the metal may be easily worked. Secondly, it is highly undesirable that to increase the amount of carbon present in the steel, as this injures the mechanical properties of the steel and increases its brittleness. The author believes more firmly than ever in the commercial future of nickel-vanadium steels, although he is now in a position to state that investigations have shown the necessity of restricting its composition within the following limits: Carbon, 0.10 to 0.30 per cent; nickel, 2 to 7 per cent, and vanadium, 0.10 to 0.30 per cent. It is quite possible that it might become necessary to reduce the amount of vanadium to 0.05 per cent.

Nickel-Silicon Steels.—The author divides these into seven classes, but these need not be described in detail in this abstract, as this series has little commercial value. The presence of silicon raises the transformation points of nickel steels, and consequently alters the percentages of nickel at which transformations from one structure to another usually occur. In this way it facilitates the transformation of γ-iron into martensite. If silicon, added to a nickel steel, permits quenching to effect a more intense influence, it must not be forgotten that this is at the expense of its resistance to shock, particularly at right angles to the direction of rolling. In short, the addition of silicon to a nickel steel is not to be recommended.

Nickel-Aluminium Steels.—The steels containing nickel and aluminium which have been examined, present, from a micrographic point of view, the same phenomena as those containing aluminium alone. When the percentage of this metal is sufficiently high the pearlite assumes a granular form, and its characteristics closely resemble those of troostite. The addition of aluminium produces, however, in nickel steels, an increase in tensile strength and in elastic limit, which has not been observed in the case of aluminium steels. The elongations decrease and finally reach very low values. It is of interest to note that aluminium does not impart any tendency to the formation of martensite, its effect being just the opposite.

Manganese-Chromium Steels.—For corresponding percentages of manganese or of nickel, the manganese-chromium steels are absolutely comparable to nickel-chromium steels. From the commercial point of view it would, therefore, appear that nickel can, in certain types of commercial steel, be replaced by manganese. This is particularly the case with the sample containing carbon, 0.250 to 0.4 per cent; nickel, 2 to 3 per cent, and chromium, 0.5 to 1.0 per cent, in which the nickel may be replaced by 1 to 1.5 per cent of manganese. This characteristic would seem to merit special investigation, which the author hopes to be able to carry out shortly.

Manganese-Silicon Steels.—Broadly speaking, the addition of silicon weakens the mechanical properties of normal manganese steels, but it confers great advantages upon quenched steels, as it induces a marked increase in the tensile strength and in the elastic limit. When the percentage of manganese is low a superior description of spring steel is met with. In no instance does the addition of silicon to a manganese steel containing γ-iron appear to be justified. From the commercial point of view manganese steels with a high percentage of manganese should contain less than 1 per cent of silicon. Manganese-silicon steels with a low percentage of manganese (i. e., below 1 per cent) are of great interest, the use of these steels for making springs being widespread.

Chromium-Tungsten Steels.—The principal conclusion to be derived from the investigation of chromium-tungsten steels have been as follows:

A. The constituents found in these steels in the normal state are pearlite, martensite and a carbide, which is apparently a triple carbide of iron, chromium and tungsten, and may be accompanied by martensite, by sorbite (or troostite) or by γ-iron.

B. The structure most frequently met with in normal chromium-tungsten steels, which is that of the rapid tool steels employed commercially, is built up of grains of carbide on a background of troostite or of sorbite.

C. With regard to the mechanical properties the chief characteristics may be summarized as follows:

(1) Pearlitic steels: tensile strength and elastic limit higher than that of ordinary carbon steels; elongation and resistance to shock lower.

(2) Martensitic steels: tensile strength and elastic limit very high; elongation and resistance to shock low.

(3) Steels containing the carbide: resistance to shock very low.

Variations in these properties depend upon the constituent which accompanies the carbide; with martensite the tensile strength and elastic limit are very high; with sorbite the tensile strength, usually moderately high (about 90 kilogrammes per cubic centimeter) varies slightly, the elastic limit is medium, as are also the elongations; with γ-iron the tensile strength depends upon the percentage of carbide present, and is high in proportion as this constituent is present in bulk, the elastic limit is low, and average elongations are obtained.

D. Generally speaking, treatment reveals nothing that has not already been noticed except in the case of quenching. The influence of the latter treatment may be summarized as follows:

(1) On pearlitic and on martensitic steels quenching has an influence which has already been investigated in several categories of steels, viz.: it transforms pearlitic steels into martensitic steels, at the same time raising in a marked manner their tensile strength and their elastic limit and lowering their elongation and resistance to shock These transformations are of much greater importance than in the case of ordinary carbon steels. The martensitic steels are slightly softened by quenching, owing to the formation of a small amount of γ-iron.

(2) Steels containing the carbide give exceedingly interesting results on quenching, varying with the temperature and the duration of heating. The higher the temperature of heating, above 850° C., the period of heating the temperature remaining constant (or the longer the period of heating the temperature remaining constant), the smaller is the amount of carbide which remains after quenching. In the steels under investigation the author was able, by varying the duration of heating, while maintaining the temperature at 1 200° C., to cause the whole of the carbide to disappear and to obtain with steels, which normally contained sorbite and carbide, a structure which would appear to be that which is required in rapid tool steels, viz.: an exceedingly fine structured martensite, almost invisible under the microscope. This proves that, commercially speaking, in order to obtain the best results with rapid tool steels they should be subjected to a definite temperature treatment and period of heating prior to quenching, which should vary with the quantity of carbide contained in the steel, and consequently with its composition.

Regarded commercially, a study of these classes of steel leads to the following conclusions:

Pearlitic steels possess very little interest for mechanical purposes. They reveal properties of less interest than nickel-chromium steels and are much higher priced. It may be, however, that they may prove useful in making races for ball-bearings or the balls themselves, owing to their extreme degree of hardness.

The only commercial application of these steels that appears feasible is for making high-speed tools, but this is, of course, of great importance, as has been clearly shown by the extra-

ordinary rapidity with which these steels have gained commercial acceptation. The investigations appear to throw a light on the conditions under which the steel can be utilized commercially as a high-speed steel. It should not in its normal state belong to any of the following groups:

Pearlitic steels, which will not do for high-speed steels.

Martensitic steels, which are too difficult to forge.

Steels containing martensite and carbide, in which the presence of γ-iron confers too great a degree of softness.

Steels for industrial use should, therefore, be sorbitic (or troostitic) or should contain carbide. The treatment they should be subjected to is such that, after annealing, the whole of the carbide should be dissolved; the rate of cooling and the composition appear to influence them but slightly; the length of time and the temperature of heating alone are of importance.

In reviewing this research as a whole, the author points out that a given set of mechanical properties corresponds with a definite structure. It is necessary to distinguish between a simple structure in which a single constituent is found and a more complex structure in which two or more constituents are present. The simple structures are martensite and γ-iron. The martensitic structure corresponds with the following physical properties:

Tensile strength and elastic limit very high.

Elongation medium, sometimes very low.

Great hardness and great difficulty in working and forging.

A γ-iron structure coresponds with the following properties:

Tensile strength fairly high, elastic limit low.

Elongation and resistance to shock usually good, although under certain conditions they may become average only.

The steels are easily forged, but are difficult to machine. Steels of a complex structure may be described as follows:

(1) All the graphite steels are unserviceable; from the moment the graphite appears the steel is brittle, and it becomes impossible either to roll or hammer it.

(2) Steels containing a carbide possess properties characteristic of the other constituent accompanying it. These properties are, however, more marked, and in particular the brittleness is increased.

With pearlitic steels it becomes impossible to foretell any of the qualities, as they depend not upon the quantity of the pearlite alone, but more particularly upon the elements, other than carbon, present. It may, however, be said that the tensile strength and elastic limit are seldom very high, although nothing can be known beforehand as to the brittleness or the elongation. Quenching usually transforms pearlite to martensite, provided it is carried out at a favorable temperature. An exception, however, must be made in regard to steels containing a fairly high percentage of aluminium. Similar treatment leaves the martensite unchanged, or simply confers upon it a tendency towards the formation of γ-iron with the resulting lowering of its properties.

Sorbite is transformed into martensite if the steel is quenched at a suitable temperature. γ-iron is not transformed on quenching, except in certain steels which are on the boundary line of the martensitic steels. With regard to steels containing carbide, the alterations induced by quenching depend upon the nature of the carbide. For the carbide of chromium steels a temperature of 1,200° C. is required for solution for carbides containing tungsten or molybdenum, even when chromium is present, a temperature of 850° C. is sufficient, but the lower the temperature and the higher the amount of carbide present, the longer should the period of heating be sustained; carbide of vanadium is never altered by quenching. Annealing does not, as a rule, alter the structure. The pearlite becomes finer; the needles of martensite better defined, and the polyhedra of γ-iron and the grains of carbide become larger. An exception must be made in the case of γ-iron steels which are on the border line of the martensitic steels. These

become transformed with the formation of martensite, while certain steels containing silicon or vanadium have their percentage of graphite increased.

Cooling and hammering are generally without influence on structures, although an exception must be made in the case of certain nickel steels, particularly those which are on the border line of martensitic steels. No matter what transformation may occur, the conclusions to be drawn as to the latter are based upon the considerations obtaining in the case of normal steels. Thus, the case of a γ-iron steel may be taken as an example. On becoming martensitic such a steel gains considerably in tensile strength and elastic limit, while it acquires a lower elongation and resistance to shock.

From these researches it would appear that the area for the commercial employment of these steels is considerably restricted, and that steel manufacturers should not depart from the limits which have been based upon the examination of the micro-structure. To begin with, the manufacturer should reject all steels having a structure which contains either graphite or martensite. It is needless to revert to the subject of the graphite; as regards the martensite, its presence creates such difficulties in working and forging the steels that it is difficult to see to what uses steels possessing this structure can be applied.

Steels with a carbide possess no interest when they at the same time contain γ-iron. It is only when they are pearlitic or sorbitic that they can be of use, and even in this case they afford opportunities for application in a few special cases only; the most interesting being for tool steels and for steels for bearings.

There remains, therefore, the two structures:

    Pearlitic steels and
    Steels containing γ-iron.

The latter can only be obtained by using high percentages of nickel or of manganese or both, to avoid obtaining steels too readily transformed by quenching, annealing and cooling. Indeed, the percentages required are considerably higher than is usually imagined. Hence the cost of manufacture is somewhat high. However, it is necessary to recall the fact that the elastic limit of these steels is very low, and that they are very difficult to work. This renders the field for their employment exceedingly limited. Thus the following conclusion is arrived at:

Setting aside steels containing pearlite and a carbide, or sorbite and a carbide, which possess considerable interest for tool-steel purposes, and, in certain special cases, for machine parts, and those containing γ-iron, which can only be resorted to in exceptional circumstances, the only type which should be sought for general application is that of steels with a pearlitic structure. This statement must be further limited by saying that steels possessing this structure and its corresponding mechanical properties should not, as a rule, contain much carbide. The field for their industrial application is thus limited to steels containing relatively low proportions of foreign elements, amongst which by far the most interesting which the author has studied are:

    Nickel-vanadium steels and
    Nickel-tungsten steels.

To which must be added, if the results obtained have been rightly interpreted,

    Chromium-vanadium steels,

which the author hopes shortly to make the subject of a separate, complete investigation.

Balance.—Under the title, "The Equal Arm Balance," CARPENTER and BISBEE published in the January issue of the *Physical Review* a paper reducing the static equations of the balance, and applied these equations successfully to two commercial balances. In the March issue of the *Western Chemist and Metallurgist*, V. H. GOTTSCHALK performs a very useful work in translating this mathematical theory into non-mathematical statements.

Two Electrochemical Processes for the Extraction of Silver and Gold.

The experiments to be described below were made by Dr. MOOSHEGH VAYGOUNY, at the University of California from 1903 to 1905, the complete account having recently been published in form of a thesis for the degree of Ph. D.

The experiments were made with some of the new well-known Tonopah gold and silver ores. The object was to find an lixiviation process for the extraction of both gold and silver, without any preliminary roasting of the ore; the solution should neither be costly nor have any properties deleterious to health; in order to avoid the necessity of its being constantly renewed, it should be regenerated; in other words, the process should by cylic.

Two independent cyclic processes were developed by Dr. Mooshegh Vaygouny.

CHLORIDATION.

The first or "chloridation" process depends upon the use of acid solutions of ferric chloride or sulphate, in presence of high concentrations of soluble chlorides, especially hydrochloric acid or sodium chloride, which act as solvents of silver chloride. Some information on this process has already been given on page 36 of our Vol. III.

The average extraction, under proper conditions, was over 95 per cent silver and from 60 to 88 per cent gold, varying with the character of the ore. The metals dissolved were recovered by electrolysis. It is of special interest that under proper conditions the re-oxidation of the ferrous salts formed during the decomposition of the sulphides, and the consequent regeneration of the ferric salts could be readily accomplished without the use of any diaphragm whatever. It was further found that there was no difficulty in carrying the re-oxidation of the solution so far as to even charge the latter with much free chlorine, which could then act as an efficient solvent of gold.

A very small percentage of glue added to the solution was found to give good-coherent deposits of the metal.

Since an account of this "chloridation" process has already been published, these few remarks must suffice.

PERSULPHATATION.

The second method depends on the use of persulphates, also in conjunction with concentrated solutions of chlorides.

In this method the chief point of interest is that both the oxidation of the sulphides of silver and of gold is brought about under the influence of one and the same compound, namely, the persulphate used, without any necessary dependence upon the generation of chlorine by electrolysis.

In bringing this double effect about, advantage is taken of the fact that persulphates decompose chlorides, setting chlorine free, ready to attack gold, especially in presence of free acids, and that, on the other hand, they attack sulphides, readily converting them into sulphates, while they themselves break down to simple sulphates.
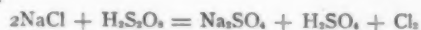
The reaction taking place in the first case seems to be of the type

$$2NaCl + K_2S_2O_8 + H_2SO_4 = Na_2SO_4 + K_2SO_4 + H_2SO_4 + Cl_2$$

the decomposition of the chloride being due to the formation, first, of free persulphuric acid, thus:

$$K_2S_2O_8 + H_2SO_4 = K_2SO_4 + H_2S_2O_8,$$
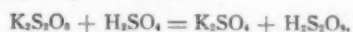
which then reacts upon the chlorides in a manner now well known:

$$2NaCl + H_2S_2O_8 = Na_2SO_4 + H_2SO_4 + Cl_2$$

While with the ordinary concentrations of acids and persulphates the attack of chlorides can be called all but rapid at best—a fact which is rather fortunate from the practical standpoint, seeing that the nature of the work to be done by the chlorine thus liberated, namely, the solution of gold, requires

only a slow but continued oxidizing effect—that of the sulphides, on the other hand, is much more readily accomplished. In fact, this decomposition goes on readily, even in neutral or slightly alkaline solutions, though it is certainly more vigorous in the presence of acids. The reactions here in question seem to take place as follows:

$$Ag_2S + K_2S_2O_8 = Ag_2SO_4 + K_2SO_4 + S$$

though much of the sulphur forming the metallic sulphides is oxidized completely over to sulphuric acid.

These being the mechanism and character of the reactions immediately involved in the use of solutions of persulphates for the solution and extraction of the precious metals, it is evident that, as with the chloridation method, so also here it is essential for the successful working of the method in actual operation to have these solutions charged with a soluble chloride. The introduction of such chlorides is, indeed, important in this case, not merely because it is necessary to have some chloride present in the solution so as to react with the persulphates and liberate the chlorine necessary to dissolve the gold in the ore, but also because AgCl is no more soluble in simple persulphate solutions than in ordinary solutions of iron salts, so that there is here the same danger of much silver being lost in consequence of the unavoidable presence of chlorine compounds occurring as impurities in ores, unless some appropriate solvent of AgCl is used here as with the chloridation method.

But there is even a more grave reason for which such a solvent of silver should be used in this connection, and that is that silver salts form an insoluble percipitate of peroxide of silver (AgO), in presence of persulphates, which is only decomposed and dissolved satisfactorily by strong chloride solutions.

In endeavoring to select an appropriate solvent for these insoluble silver compounds, it was at first thought that ammonia would be a desirable substance to use in conjunction with this method, inasmuch as it would act as a very efficient solvent of any silver chloride which might form in the course of the treatment, while, at the same time, the action of persulphates upon the sulphides would not thereby be impaired in any way. This belief, had. however, soon to be abandoned.

It was, indeed, observed that, far from being a suitable substance for the purpose, ammonia was decidedly objectionable, inasmuch as it was found to be rapidly decomposed into free nitrogen and water whenever any appreciable amount of silver and a persulphate were found in solution with it. This oxidation of ammonia under these circumstances is due, doubtlessly, to a catalytic reaction depending upon the formation, primarily, of the peroxide AgO, which then attacks, more or less violently, any free ammonia which may be present in the solution, and causes its complete decomposition in a well-known manner.

The reaction is, indeed. so characteristic that Dr. M. Vaygouny has found it to be a very handy way for detecting the presence of silver salts in chloride solutions, which is the more useful, as it is often impossible to detect silver in such solutions whether by any other simple precipitation method or by a mere dilution of the solution, with the hope of thus diminishing the solvent power of the liquid on AgCl, and thereby causing a turbidity of this compound to appear. In carrying out this test, a strong solution of a persulphate, preferably the ammonium salt, is added to the solution to be tested, and then this latter is made strongly alkaline with NH₄OH, when, if silver be present, a more or less rapid evolution of nitrogen gas will be observed to take place.[1]

The use of ammonia as a solvent of insoluble silver salts being thus out of question, and none of the other non-chloride solvents being sufficiently stable in presence of persulphates. the choice of concentrated solutions of a chloride thus becomes a necessity.

Now, in the first place, persulphates appear to be good re-

agents for treating ores of the precious metals, because they are good oxidants, capable of being regenerated by the electric current; further, though a small percentage of a chloride in concentrated solutions of sulphates is very advantageous for the electrolytic formation or regeneration of persulphates,[2] higher percentages of chlorides are quite injurious to such regeneration. The question may, therefore, arise here as to whether the means above stated for holding silver salts in solution in actual practice would not render the process as a whole inoperative, at least as a "cyclic" process. It is, indeed, only too evident that, an essential condition for the formation of persulphates by means of the electric current being a high rate of discharge, of SO₄ ions at the anode, the introduction of unduly large proportions of a non-sulphate electrolyte, such as NaCl. will tend to diminish the yield of the current in persulphate formation. How this difficulty may be overcome in practice will be discussed further on.

In comparison with the chloridation method the following advantages of the persulphatation process may first be mentioned:

1. Theoretically the use of a persulphate should be preferable to that of ferric salts alone, because, unlike the latter, whose reaction products—the ferrous salts—are notorious for their tendency to reprecipitate the precious metals from their dissolved state, persulphates break down merely to sulphates, that is, salts having no such injurious effects upon the solutions of these metals.

2. For this reason, principally, simple gold ores could be treated by persulphatation to a better advantage than by chloridation, or perhaps by any other wet method.

3. In practice one may have to treat ores that are too rich in limestone to be economically handled by the chloridation process, inasmuch as they would first have to receive a preliminary acid treatment before they could be subjected to the action of ferric salts, lest these latter should be precipitated out and the process rendered more or less inoperative. Such ores should be amenable to persulphatation to a better advantage, seeing that persulphates can decompose sulphides of silver in neutral or slightly alkaline media as well as in acid.

On the other hand, with very "base" ores, persulphatation cannot be expected to be nearly as useful a method as chloridation, owing to the fact that the satisfactory oxidation of such ores by wet methods such as these. requiring, as they do, the heating of the reacting solutions, persulphates would be rapidly decomposed under these conditions, and hence be useless as economic agents for treating this type of ores.

The persulphate used by the author was the potassium salt throughout; for the reason that the ammonium salt is rather unstable, while sodium persulphate could not be had on the market in California nor can it be made easily or readily in a solid condition.

Both the decomposition of sulphides and the solution of gold can be best brought about by persulphates when some free acid is present. Moreover, more or less ferric salts are introduced into the solution through the agency of the acidity of the liquids upon the compounds of the iron which may be in the ores, and hence the oxidizing power of the solution is thereby enhanced.

Comparative experiments of the author show that the chief function of the acid lies in that it facilitates the solution or extraction of the gold rather than that of the silver values. On the other hand, the variation in the concentration of the persulphate in solution effects especially the extraction of silver alone. From the different tests of the author it appears that the best conditions for the process involve the use of a solution containing 2 per cent of persulphate of potassium and 5 per cent of sulphuric acid with 20 per cent of sodium chloride.

An intermediate leaching of the ore masses with a solution of an alkali, such as sodium hydrate, facilitates the further

[1] This catalytic reaction has also been studied and similarly explained by Hugh Marshall (J. Chem. Soc., 1901, Abstr. ii., page 156).

[2] Elbs, Zeitsch. für Elektrochemie. 1895, page 245.

attack of the sulphides. This was found to be due to the fact that such alkalies dissolve away the sulphur and help to remove this element, which otherwise forms a coating around the particles of the sulphides, and thus hinders their further decomposition. The reaction giving rise to the formation of free sulphur may be represented thus:

$$Ag_2S + K_2S_2O_4 = Ag_2SO_4 + K_2SO_4 + S$$
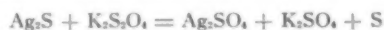
That the beneficial effect of NaOH is in reality due to its power to merely dissolve considerable amounts of spongy sulphur and help to remove it out of the ore mass, thus laying bare the remaining particles of sulphides subject to the further attack of the main oxidizing solution, is proven by the fact that the now impure, brownish solution of NaOH, recovered after the treatment of the ore with it and after filtration of same, gave a decided precipitate of yellow, flocculent sulphur when neutralized with H_2SO_4, along with a little H_2S.

Some experiments were also made with alkaline ores so as to determine to what extent such ores might be treated with the persulphatation process without necessitating the preliminary neutralization of their carbonates with correspondingly large amounts of acids. With respect to the treatment of such ores the persulphatation process has a decided advantage over the chloridation process.

As has already been mentioned above, the regeneration of the persulphate solution presents difficulties, while the electrolytic recovery of the precious metals from the sulphate solutions is as easy as in the chloridation process.

The difficulty of the regeneration of the persulphate solution is due to the presence of the large amounts of chlorides. To work the process on a large scale it would seem to be necessary to remove part of the sulphate in solution from an aliquot part of the latter, conferring it into a persulphate in a special electrolytic cell (containing no chlorides), and finally returning it to the main solution as needed.

There are several methods of solving this problem. If the sulphate worked with in practice be the potassium or ammonium salt, part of the liquors holding this latter in solution may be shunted off—best directly after the recovery of the precious metals—and a desired amount of the sulphate made to separate out in the solid form, either by concentrating the solution with respect to this salt by evaporation, or by cooling it sufficiently, according to practical facilities. In either case, the solution being more nearly saturated with respect to the sulphate than the chloride present, the former salt will naturally separate out first, in form sufficiently pure and free from admixed chlorides to be redissolved in a fresh, sufficiently chloride-free solution of the same sulphate, and electrolyzed in special electrolytic cells, with a view to converting it, at least in part, to persulphate, which may then be removed from the solution and returned to the stock liquors.

While, with potassium sulphate chosen as the starting material to work with in practice, the evaporation method seems to be much more likely to be practicable, with the sodium salt (Na_2SO_4), on the other hand, the separation of the solid Na_2SO_4 from the shunted solutions by cooling seems to be equally likely to meet with success in practical operations. Indeed, the very rapid decrease of solubility of the Na_2SO_4, with the lowering of the temperature of its solutions, and the very slight decrease under the same conditions of the solubility of NaCl, which may be present in the same solution, render this method highly useful as a means of bringing about the separation of the former salt from a concentrated solution of both these compounds in a form sufficiently free from the latter for all purposes concerned, and that with only a slight lowering of the temperature of the solution.

Troubles may be experienced in the latter method, due to the difficulty of preparing the persulphate of sodium in solid form owing to its excessive solubility. The author thinks that it is not absolutely necessary to prepare the persulphate in

the solid form, but that the sulphate of sodium (whether original or recovered from the shunted parts of the stock solutions) may be dissolved in water and then converted, to any desired extent, into a solution of the corresponding persulphate, which solution may then be returned, in body, to the main or stock liquids.

German Bunsen Society—Report of Dresden Meeting—II.

By H. DANNEEL, P.H. D.

NITROGEN OXIDES FROM AIR.

The various papers which were presented on the subject of the fixation of atmospheric nitrogen (and a digest of which was given in our July issue, p. 256) elicited considerable discussion.

Prof. Biltz asked whether it would not be practical to try and produce nitrogen pentoxide. Its temperature of decomposition is very low, and a catalytic process could be easily applied, in order to get a high efficiency and high reaction velocity. N_2O_5 dissolves easily in water, forming HNO_3.

Prof. Foerster stated that a mixture of steam, NO_2 and O_3 easily reacts, forming nitric acid.

Prof. Warburg said that it is easy to absorb the dilute NO_2 gases by KOH or NaOH, if some O_3 is added to the gases. Prof. Nernst called attention to the fact that this might yield a very suitable analytical method for testing the dilute NO_2 gases.

Prof. Helbig stated that N_2O_4 in liquid as well as in gas form is easily oxidized by O_3 to N_2O_5. It seems, therefore, that the effect of O_3 is not catalytic, but a simple oxidation.

Prof. Haber doubts whether an output of 900 kg. HNO_3 per kilowatt-hour has really been obtained on a large scale. In general it will hardly be possible to get more concentrated gases than 2 per cent, and the efficiency will scarcely be higher than 600 kg. HNO_3 per kilowatt-hour. Prof. Klandy replied that 5 per cent gases and an efficiency of 900 kg. have probably been obtained in the laboratory on a small scale only.

Dr. Erlwein, of the Siemens & Halske Co., then showed furnaces and apparatus for the production of calcium cyanamide. Ground calcium carbide is placed in retorts heated to 500° or 900° C. Nitrogen is then introduced and is quickly absorbed by the carbide. The cheapest method of producing the nitrogen is by means of liquid air. At the cyanamide plant in Piano d' Orta, Italy, the cost of producing the nitrogen from liquid air is 0.75 to 1 cent per kilogram, and 330 kg. of bound nitrogen are obtained per horse-power-year.

Dr. Kraus called attention to a patent of Polzeniusz, who mixes 10 to 15 per cent of $CaCl_2$ with the carbide. This mixture absorbs nitrogen at a temperature of 700° to 750° C., while pure carbide requires a temperature of 900° C. Dr. Arndt thinks that this effect of $CaCl_2$ is due to the fact that it is partially dissociated already at 700° C., while Prof. Foerster and Prof. Nernst believe that the molten $CaCl_2$ acts here as solvent, so that the reaction proceeds more easily. This is evident, for instance, from the fact that only such additions are effective, the melting point of which is below that of carbide, for instance, $CaCl_2$ and CaF_2, but not $CaSO_4$.

In reply to a question of Dr. Goldschmidt, Dr. Erlwein stated that ordinary cheap commercial carbide may be used, and that there is no need of taking expensive brands.

Prof. Hempel called attention to the large quantities of nitrogen wasted in water-closets. A problem of engineering of the future will be to find a method which combines the cleanliness of water-closets with a possibility of preserving the nitrogen compounds. Prof. Haber called attention to the possibility of utilizing the coke oven gases.

Explosives.

Prof. WILL then read a paper on technical methods of testing explosives. For the efficiency of an explosive, not only the heat which is developed is of importance, but also the reaction velocity. The pressure of explosion is a measure of the operating capacity of an explosive. The author gave a review of various technical methods for testing explosives, and pointed out that to get a correct idea of the value of an explosive several methods must be employed.

Standard Cells.

A paper by Dr. H. von STEINWEHR, of the German Reichsanstalt, discussed the influence of the size of the grains of Hg_2SO_4 on the e. m. f. of the Clark cell. Thomson has first shown that the vapor tension of small drops of water is larger than that of greater surfaces of water. Ostwald has applied this to the solubility of salts, and has found, for instance, that fine HgO is more soluble by several per cent than coarse HgO. The e. m. f. of a combination Hg, fine HgO, coarse HgO, Hg may amount to 0.6 millivolt. Hg_2SO_4 behaves in a similar way In order to get constant standard cells it is necessary to use very coarse Hg_2SO_4.

Catalysis.

Prof. G. BREDIG spoke on catalysis in heterogeneous systems and a new mercury oxide. Platinum black and colloidal platinum in solution accelerate greatly the reaction $2H_2O_2 = 2H_2O + O_2$. If used often the platinum becomes less active, and cannot be rejuvenated. Platinum also loses its efficiency if "poisoned" by certain reagents like KCN. This poisoning can be counteracted by purification. Platinized platinum sheets ("macroheterogeneous catalysis") behave just like a colloidal platinum solution ("microheterogeneous catalysis"), if we neglect small differences. In both cases Nernst's law is fulfilled, according to which the reaction velocity is due to a diffusion phenomenon, the diffusion drop being within a thin layer in the catalyser. By experiments of Nernst and Brunner this layer has been found to be 0.02 mm. thick. The same value also follows from the experiments of the present author. Metallic mercury is also a catalytic agent accelerating the decomposition of H_2O_2. A brown oxide of the formula HgO_2 is formed. It is strongly explosive.

Allotropic Forms of Gold and Silver.

Prof E. COHEN discussed this subject. According to Thomson, there exist three allotropic forms of gold, as has been concluded by him from calorimetric measurements. According to Berthelot, there exist five allotropic forms of silver. The present author argues that different allotropic modifications of the same metal should differ at least in their free energies. He has prepared these alleged different modifications exactly according to the instructions of Thomson and Berthelot, and has measured their potentials against each other. No difference was found. The author concludes that the "different" forms are identical.

Decomposition of Nitrites.

Prof. R. ABEGG discussed the decomposition of nitrites. Silver nitrite, $AgNO_2$, when heated decomposes, according to the formula

$$2AgNO_2 = AgNO_3 + NO + Ag,$$

or written as ionic reaction

$$Ag \cdot + 2NO_2' = Ag + NO_3' + NO$$

The reaction is completely reversible and the equilibrium depends on the NO pressure. One NO_2' ion robs the $Ag \cdot$ ion of its positive charge, and simultaneously gives off one O to another NO_2' ion. We have, therefore, a real oxidation equilibrium, and the energy of the reaction may be found by measuring the e. m. f. of the combination Ag, Ag solution, solution of $NO_2 + NO_2 + NO$, Pt. The oxidation potential

of the reaction $2NO_2' \rightarrow NO_3' + NO$ must be more positive than the potential of the reaction $Ag \cdot \rightarrow Ag$, if the process can go on at all.

The potential of the latter reaction is — 0.77 volt, and that of the former reaction was found to be — 0.5 volt, so that the decomposition goes on with an e. m. f. of 0.27 volt, if all substances participating in the reaction are present with the concentration 1.

In general it follows that all those nitrites will decompose, the metal of which is more noble than — 0.5 volt, f. i. platinum nitrite, gold nitrite, etc. The e. m. f. of decomposition changes, of course, when the concentrations change. If, for instance, complex salts are formed in the solution which remove the NO_2' ions, a nitrite with a metal nobler than — 0.5 volt may not decompose. On the other hand it is possible to force a nitrite with a non-noble metal to decompose according to the law of mass action, if one removes the reaction products; for instance, if one takes care to remove NO continually from the solution.

Alkali metal nitrites are stable. Their solution also contains the ion H, which may be reduced according to the reaction

$$H \cdot + 2NO_2' \rightarrow H + NO_3' + NO$$

The reaction of the H ions proceeeds easily if the H_2 gas and the NO gas are continually removed from the solution. For this reason the alkali metal nitrites, when left in the open air, must decompose with development of H_2 and must become alkaline; the oxygen in the air continually oxidizes any NO which is formed to N_2O_3. This is a well-known fact.

Radiation Laws.

Prof. LUMMER, of the German Reichsanstalt, gave a very interesting review of recent progress in the theory of radiation. It is, of course, impossible to abstract such a paper. The author distinguished two kinds of radiations. First, rays which simply transport energy from the radiating body; besides light rays, Roentgen rays also belong to this class. Second radiations in which simultaneously material particles are thrown off; radium rays and cathode rays are examples of this class. Electrons always participate in radiations. Thus ultraviolet light dissipates electric charges. The author then gave a review of the corpuscular, or electronic, theory of the constitution of matter, and discussed the production of the spectrum of an element from this point of view.

Ammonium.

Prof. A. COEHN spoke on ammonium. The metallic radical NH_4 has not yet been prepared, since it was found impossible to fulfill the conditions under which it will be stable. On the other hand, it is possible to produce electrolytically ammonium amalgam which, however, is decomposed when no longer subjected to the influence of the cathodic potential; during the decomposition NH_3 and H_2 are formed. The decomposition is accompanied by an evolution of energy, and since it is somewhat similar to the disintegration of radium it did not seem impossible that radiations would be produced in this case, just in the case of radium. Prof Coehn endeavored to prove experimentally that such radiations occur.

The mercury cathode, which was contained in a glass vessel (somewhat similar in shape to the small type of German smoking pipe), was placed in an NH_4Cl solution and electrolyzed for 3 minutes with constant current. The electrode was then quickly removed and placed below a metallic disc. When this disc had previously received a negative charge this electrostatic charge decreased, on account of the proximity of the disintegrating amalgam. When the disc had not been previously charged it assumed a positive charge, and the amalgam remained negatively charged. A very small quantity of amalgam only is required to produce this effect. If only 0.0004 gram NH_4 was in the amalgam the electrostatic effect was evident. With potassium or sodium amalgam no such effect was observed.

The author has not directly proven that this is a radiation phenomenon, due to the disintegration of the amalgam, although it seems so.

COLLOIDS.

Three papers relating to this subject were presented. Dr. A. LOTTERMOSER gave a very concise and complete summary of the whole subject of hydrosoles, Dr. SIEDENTOPF spoke on colloidal alkali metals. He also exhibited the new "ultra-microscope."

Prof. R. ZSIGMONDY discussed the size of the particles in hydrosoles. The study of hydrosoles has been greatly furthered in recent years by the invention of the "ultra-microscope." In some hydrosoles the particles can be directly seen under the microscope. These visible particles the author calls submicrons, while those particles which are no longer visible are called amicrons.

The author determined the size of the latter particles in a very clever way. Such small gold particles may be used as a catalytic agent to disturb the equilibrium of an over-saturated silver solution. The silver then crystallizes around each gold amicron, so that the different amicrons get bigger in size and can now be counted under the microscope.

Preliminary experiments on the size of the microns gave dimensions of 1.7 to 3 $\mu\mu$. By the way, it may be mentioned that 5 $\mu\mu$ is the order of size of large molecules like starch molecules, which have a molecular weight of 30,000. The submicrons are seen under the microscope to be in continual motion.

PASSIVITY AND CATALYSIS.

Dr. O. SACKUR presented the following picture of the anodic solution of metals. A metal tends to get into equilibrium with the hydrogen ions which are always present in the solution, according to the formula

$$2Me + 2H \cdot = 2Me \cdot + H_2.$$

or with bivalent metals, according to the formula

$$Me + 2H \cdot = Me \cdot \cdot + H_2$$

On account of the anodic polarization oxygen ions discharge and remove the H_2, forming water according to the equation

$$2H_2 + O_2 = 2H_2O$$

Since the hydrogen has now disappeared, metal will again pass into solution to get into equilibrium with the hydrogen as described above.

If there is any reason why this cannot take place the metal will no longer dissolve, and the metal is then in the passive state. Now, we know that the formation of water from $2H_2$ and O_2 is very slow if there is no catalytic agent present. Therefore, every metal which has not an accelerating catalytic effect on the formation of water must assume the passive state with anodic polarization. Instead of the solution of metal we then get liberation of oxygen.

If we, therefore, arrange the metals in a series according to their ability of assuming the passive state, and in another series according to their accelerating catalytic effect on the formation of water, then these two series must be essentially the same. The author has measured the catalytic effect of the metals in water formation by "reststrom measurements," and has found that the parallelism sketched above really exists. Passivity is, therefore, intimately connected with the reaction velocity.

DISSOCIATION THEORY.

Prof. DUTIOT spoke on molecular conductivity and the laws of dissociation in organic and inorganic solvents. He referred to Prof. Kahlenberg's criticisms of the dissociation theory, emphasizing the difficulties which the dissociation theory has to meet with solvents other than water. There are apparently a number of organic solvents in which, for instance. the law of Kohlrausch of the independent migration of ions does not hold true.

The author has made extended experiments with various solvents and various concentrations, and found in nearly all cases a very great influence of light on the result. If the effect of light is excluded, or numerically taken into account, he finds that solutions in solvents other than water behave exactly like aqueous solutions. The behavior is entirely analogous to that of aqueous solutions. The law of Kohlrausch of the independent migration of ions was found to hold good in all solutions which were investigated. The law of mass action (the dilution law of Ostwald) holds true in dilute solutions.

Besides the papers abstracted above a number of papers were presented which were of purely chemical interest.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)
THE FARADAY SOCIETY.

A rather sparse attendance of members marked the ordinary meeting held on June 12. (A preliminary account of which was already given in our July issue, page 275.) Mr. Murray Morrison presided. The first two papers (a), "The Electrolytic Deposition of Zinc-Using Rotating Electrodes," by Dr. T. Slater Price and Mr. G. H. B. Judge, and (b) "A Simple Form of Rotating Cathode for Electrochemical Analysis," by Dr. F. Mollwo Perkin, were read in abstract by Dr. Perkin, with a running comment and a series of blackboard drawings.

In regard to the second paper, some later figures were given as to the rate of deposition on nickel cathode. With .1266 grammes of copper present in a solution, .1259 grammes were deposited in 40 minutes. With 0.875 grammes of iron present, apparently 0.895 grammes were deposited in 35 minutes, this discrepancy being due to the deposition of carbon at the high current density of 1 amp. per square decimeter. As a cathode material nickel was just as good as platinum, nitric acid successfully dissolving off a copper deposit. sulphuric acid successfully dissolving off iron deposits. Iridium anodes, although costing 150 per cent more were recommended in place of platinum.

These two papers being discussed jointly, Dr. Borns asked where the carbon came from. Dr. Perkin replied that it was due to the presence of oxalate or tartrate. Dr. Walker suggested that the throwing over of water with some rotating electrodes was evidence to the effect that electrochemists had often failed to appreciate the importance of studying mechanics. He also asked whether the iron deposit mentioned by Dr. Perkin had been examined micrographically. Mr. Gaster asked whether the lubricating oil used for the vertical spindles got into the electrolyte.

Mr. Spiers mentioned some experiments in which he had been engaged in the deposition of zinc for accumulator purposes. Very fine, smooth deposits of zinc were obtained on rotating cathodes running from 800 to 1,000 r. p. m. The results he had noted were not quantitatively accurate, but the current efficiency was over 90 per cent. The electrolyte consisted of saturated zinc sulphate, to which 7 per cent of sulphuric acid was added. In reply, Dr. Perkin pointed out to Dr. Walker that it would be extremely difficult to etch cathodes with such thin deposits. There would probably be some interesting information available on this subject, as Mr. Cowper-Coles was carrying out some large-scale experiments on the deposition of pure iron.

The third paper, by Mr. S. Binnings and Dr. Perkin, entitled "The Electrolysis of Solutions of Throcyanites in Acetone and Pyridine, Part I.," had not been circulated among the members. It was perhaps on this account that Dr. Perkin's interesting little lecture was followed with close attention. As to whether the electrolytic product was similar to canarin was a little obscure. In some respects the evidence was adverse, as

when boiled with pyridin the electrolytic canarin blackened. The product from the electrolysis of the acetone had a more orange color, but gave the same reaction on boiling. Using a platinum anode solution of 20 per cent of ammonium throcyanide. an e. m. f. of 50 volts was required for a current of 5 amps., but with a rise in temperature the voltage fell to 3.5, thus increasing the watt-hour efficiency. At the higher voltage the current efficiency was 25 per cent. The product being an anode one, it was at once found that the anode became coated, and the flow of current ceased. The anode was, therefore, kept clear by brushing, bristles being fixed in the revolving cathode. The curious results found with using lead anode, when lead throcyanide was, were also mentioned.

In conclusion, Dr. Perkin confessed the reason for the reading of the first part of a paper which had not yet been circulated. He and Mr. Binning had been at work on the subject for some months. They thought it well to put their present conclusions on record, because they had learned that Dr. Kahlenberg was at work on pyridin solutions, with a view to finding evidence tending to discredit the ionic hypothesis. Dr. Kahlenberg would be certain to stumble on this, and the authors wished to be smart enough to "get there" first.

A very brief discussion followed. Dr. Perkin answering Dr. Borns, told of a dog which died after receiving three doses of canarin, amounting to 12 grammes, with its food in three days. Mr. Digby asked how the current efficiency was affected by the rising temperature. The falling voltage had given an increased watt-hour efficiency; had the current efficiency remained constant? Dr. Walker retorted on the previous speaker by pointing out that the current efficiency depended upon the resultant product, and that it was not certain what the resultant product was.

After the usual votes of thanks the meeting then adjourned.

Exhibits at the Royal Society's Conversazione.

Very few of the exhibits at the Conversazione held on June 20 were of any electrical or metallurgical interest. Three, however, call for mention in these notes.

Dr. T. G. Moody exhibited specimens illustrating the indifference of oxygen towards iron in presence of water, and the effect of the admission of carbonic acid. A specimen of Swedish iron remained perfectly bright after exposure for three weeks to water and 32 liters of air feed from carbonic acid. At the end of the period named, air, cleaned by its passage through a tower packed with moist pumice stone, but containing its normal amount of carbonic acid, was admitted.

After 72 hours, during which time approximately 16 liters of air passed through, considerable rusting had occurred.

Mr. R. G. Durrant exhibited tubes, diagrams and experiments to show that ionic separation occurs when solutions of acids or of salts are allowed to diffuse into sensitized jellies or solutions. Acids when they diffuse into blue litmus jellies produce a purple zone in advance of the diffusion front. and (where the acid has a bleaching union) a yellow-red zone behind the diffusion front. An acid mixed with litmus on diffusing, leaves the litmus behind. When concentrated silver nitrate solution diffuses into purple litmus, under suitable circumstances, red, blue, colorless and gray zones—corresponding to H, OH, NO and Ag, appear in this order. At certain periods sharp bands may form, whose boundaries, if measured from the diffusion start, almost exactly correspond to known mobilities of these ions.

Dr. F. D. Chattaway exhibited copper mirrors obtained by the deposition of metallic copper on glass. The method of silvering glass by depositing the metal in a thin film by reduction of some soluble silver compound has long been employed in the production of mirrors, but, hitherto, no method of similarly depositing copper in a brilliant film has been discovered. The exhibit showed a number of glass vessels on which copper had been deposited by a low reduction of the black oxide.

Market Prices During June.

The chief feature of the month has been the lower prices and unsteady condition of the copper and tin markets. From the 1st until the 13th copper was exceptionally firm and steady at about £86 per ton. By the 19th the price had receded to £82.5.0, and although there was a recovery on the 21st to £83.17.6, a further decline followed, with the result that £81.5.0 was reached by the 26th, and closed on the 30th at £81.7.6. Tin opened in the neighborhood of £180, and continuing to fall reached £175 on the 14th; prices rallied on the 22d to £180.10.0, subsequently declining to £177 for cash. English lead has been fairly steady, large quantities changing hands, the prices varying between £16.15.0 and £17.5.0 per ton. Cleveland pig iron rose from £2.9.6 on the 1st to £2.11.0 on the 11th, being slightly irregular subsequently. On the 30th the price closed at £2.10.0. Antimony is quoted at £120 per ton, and quicksilver at £7.1.0 to £7.5.0 per 75-pound flask.

Shellac is quoted at £10.5.0 per case, and the Calcutta corner seems a strong one. Ammonia sulphate fetched £11.16.3 on the 30th. The coal tar derivatives and alkaline products are practically unchanged.

London, July 4. 1906.

·ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS

Electric Furnaces.

Induction Furnace.—T. F. Snyder, 825,359, July 10. Application filed July 15, 1904.

The construction of the induction furnace is shown in Fig. I. The core a, which is made of iron laminations or strips clamped together at the corners, is arranged to pass horizontally through the wall b of the furnace. This core is in the shape of a rectangle, and the two horizontal arms which pass through the furnace are provided with a series of water-troughs c set at intervals between the laminations. The water is passed through for cooling purposes, d being the inlet pipe and e the discharge pipe. Primary coils f are wound upon the core a on either side of the furnace, and are electrically connected with the generator g. The furnace is built up around the magnetic core in such a way as to provide a melting chamber g, which is provided with a reverberatory roof g' having charging openings g^2g^3 and a vent h for the escape of gases.

A bridge k is built across the central portion of the furnace from wall to wall, immediately surrounding the horizontal core a, and dividing the vertical chamber into two vertical wells on either side of the bridge. A number of bars of copper m are set in the furnace across the bridge. When molten material is introduced into the chamber g above the level of the bridge, a complete secondary circuit is thus formed around the core a and in inductive relation to it. The vertical wells on either side of the bridge k are filled with molten metal, slightly below the top of the bridge and forming molten "electrodes" united at the bottom by the copper bars m. Slag is provided in the chamber resting on the surface of the molten metal and filling the chamber to a level above the surface of the bridge, so as to close the electrical circuit. The material to be smelted is fed through the openings g^2g^3 upon the surface of the molten slag, and is gradually reduced; the metal which is liberated sinking into the wells on either side of the bridge. The tap holes l

and o for the slag and the metal also serve as a means for regulating the resistance, since when the level of the molten material above the bridge k is lowered the cross-section of the secondary circuit is reduced at this point, thus increasing the resistance. The secondary circuit, which is arranged in a vertical plane, as shown, is composed of conducting materials having different specific gravity—that is to say, the heavier material at the bottom should be a good conductor, while the lighter material at the top should be of comparatively high resistance. Since the heating effect of an electric current increases in proportion to the resistance of the conductor through which the current is passed, it will be seen the maximum heat-

FIG. I.—ELECTRIC INDUCTION FURNACE.

inng effect of the current in the secondary circuit will be developed at the top—that is to say, in the slag upon which the ore to be smelted is fed. The metal of the molten electrodes is a comparatively good conductor, but not so good as the copper bars at the bottom, so that those electrodes will be heated to a sufficient extent to keep them molten. In the smelting of iron the copper at the bottom may be molten if desired. It will remain at the bottom by virtue of its specific gravity. The slag, on the other hand, will always be light enough to remain at the top, the ore to be smelted in turn floating upon the surface of the slag.

Low-Carbon Ferro-Alloys.—E. F. Price, 825,348, July 10, Application filed Nov. 12, 1905.

The process depends on the possibility of producing a high-percentage ferro-silicon, low in carbon, and on the subsequent use of this ferro-silicon as a reducing agent. The process is a continuous operation comprising two stages. In the first stage, ferro-silicon high in silicon and low in carbon is produced by electrically smelting a charge of silica, iron ore (or iron) and carbon. The molten silicide is tapped from the smelting furnace and allowed to solidify. The ingot is then broken into fragments, which are mixed with a granular body of the compound to be reduced—for example, chromite—and the mixture is melted in an electric arc furnace. A basis flux, such as lime, is mixed with the charge to convert the silica produced by the reduction of oxide ores into a fusible slag. The furnace is operated continuously, the metal and slag being drawn off through tapping holes at different heights and new charge is added to the furnace.

Calcium Carbide.—E. F. Price, 826,742, 826,743, 826,744, 826,745, July 24. Applications filed Oct. 13, 16, 1903, and April 4, 1904. (Assigned to Union Carbide Co.)

The charge, which is a mixture of finely-divided lime and coke, is showered downwardly through a preheating chamber in which it is subjected to the action of a flame and heated gaseous products of combustion. It is then collected or massed into a body in an electric furnace, and heated to the required temperature, either by passing an electric current through the heated charge or the molten products acting as a "resistance

conductor," or through a separate resistance conductor in proximity to the charge.

The preheating is effected by withdrawing the waste carbon monoxide from the electric furnace and burning it in the preheating chamber. Other fuel, such as natural gas or oil, may be employed as a substitute for or adjunct to the carbon monoxide.

Electric Furnace.—John F. Hammond, 825,386, July 10. Application filed Sept. 18, 1905.

The furnace comprises a muffle, in the interior walls of which heating coils are variously disposed and connected, so as to form a plurality of groups. By means of a special switch or commutating device the electric current can be made to energize the heating coils in various combinations. For instance, first all coils may be in series, then the different coils may be connected in different series-parallel connections, and finally all groups are in parallel. The object is to have full control of the temperature, and especially to accomplish that the increase of temperature of the furnace is very gradual. The construction of the commutating device is described in detail with diagrams of the different methods of connection.

Electric Dental Furnace.—H. Platschick, 826,962, July 24. Application filed Nov. 27, 1905.

Details of construction of an electric muffle furnace, in which the muffle is easily detachable from its casing, from the electric wires and also from the prometer. "More pariticularly, the invention consists in combining in the muffle itself with the thermo-electric couple, which is usually constituted by different alloys based on platina, suitable resistances so adjusted as to compensate for the differences which may exist between the electric and thermic constants of divers couples of the same make by reason of the irregularity of the alloys. This arrangement allows of delivering over to the trade muffles which are interchangeable—that is, all the couples present between their terminals the same thermo-electric constants, and which consequently all give in connection with one and the same galvanometer the same pyrometric indications when they are subjected to equal temperatures. The necessity of a fresh standardizing or of a particular adjustment when a muffle has to be replaced is thus avoided, the result being a great saving of time and money."

Resistance Furnace.—I. L. Roberts, 821,830, May 29, 1906. Application filed May 31, 1904.

The construction is shown in Fig. 2, the upper and lower diagrams being a plan and vertical section respectively. A is an iron jacket, B a heat-insulating material, like a mixture of asbestos and magnesium carbonate, ground together

FIG. 2.—RESISTANCE FURNACE.

and well packed in place. The chamber C is built up of refractory material, like fire-brick. Within the same is supported the crucible D of conductivity material, like iron. Between the walls of the chamber and the vessel is a granulated resistor, like granulated or powdered coke, mixed, or not, with sand or magnesia or lime. This is in contact with the terminal or "electrode" plates F F. In a modification of this construction, the crucible rests on the layer of granulated resistance material, and the crucible itself forms the other terminal.
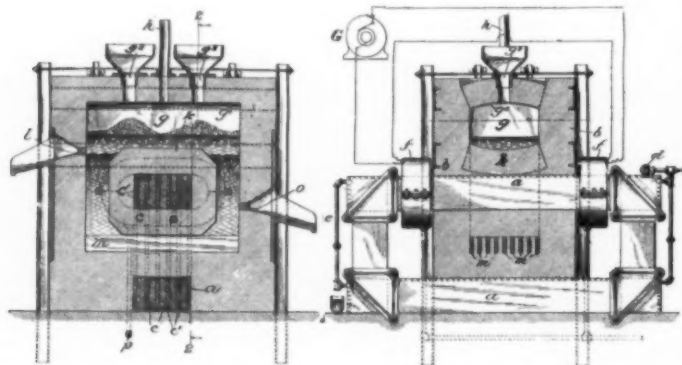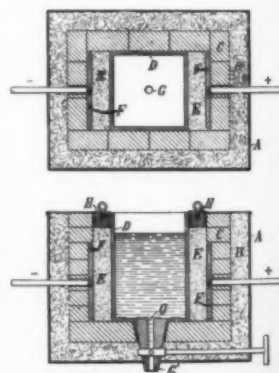
Rotary Electric Furnace.—W. McA. Johnson, 825,058, July 3, 1906. Application filed Oct. 13, 1903.

This rotary furnace is particularly designed for the reduction of zinc ores and the distillation of the metal. The construction is shown in Fig. 3. The furnace consists of a plurality of transverse sections 1 of cast iron and cast-iron end plates 2. Each section 1 is provided with a lining 3 of refractory brick. In assembling the furnace, gaskets or plates 5 of asbestos or other refractory insulating material are clamped between the several sections and between the·end sections and the end plates 2. In this manner an air-tight structure is provided, composed of several independent and electrically-insu-
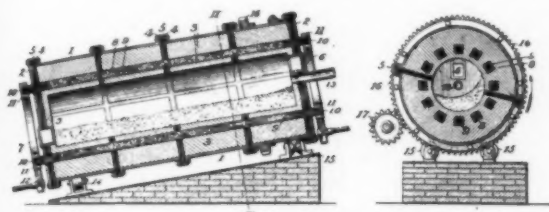
FIG. 3.—ROTARY ELECTRIC FURNACE FOR ZINC DISTILLATION.

lated parts, whereby loss of current by leakage through the furnace walls between the terminals is reduced to a minimum. A plurality of longitudinal passages 8 extend through the refractory lining. They are filled with powdered coke or graphite, which acts as resistor, terminal connections being provided in form of the graphite plugs 10. The zinc vapors are led off through pipe 13 to condensers.

Calcium Carbide Furnace.—J. E. Hewes, 821,574, May 15, 1906. Application filed July 18, 1905.

The charge of lime and carbon is placed in trough-shaped hearths, and subjected to the action of an arc between two electrodes arranged at an angle to each other. The hearth is slowly moved past the electrodes, so as to remove the formed carbide from the zone of fusion immediately after it is formed. The process is thereby rendered continuous. For this purpose the trough-shaped hearth plates are placed on two endless traveling chains or carriers. As the chains are moved along, additional hearth plates are added from time to time to the feeding end, and corresponding plates with the product are removed from the distant end of the series. This forms the traveling hearth. The two electrodes are at a right angle, one being vertical and the other horizontal, the vertical electrode being at a higher elevation than the horizontal electrode. In starting the furnace, the vertical electrode is lowered to the horizontal electrode, so as to close the circuit. The horizontal electrode is then drawn slowly backward, so as to approximate 8 inches from the vertical electrode, and then the vertical electrode is slowly raised. The height to which it is raised (together with the speed of the traveling·hearth, determines the thickness of the carbide formed.

Calcium Carbide.—John M. Morehead, 825,234, July 3. Application filed March 24, 1905.

This patent refers to the process of manufacturing carbide in the electric furnace, described and illustrated in our Vol. III., page 153. In the first claim, the inventor claims "the process of producing calcium carbide, which consists in establishing a region of reduction under a gas-tight hood, feeding the charge into and downward within the hood into the region of reduction, sealing the hood by a layer of the charge material extending below the lower end of the hood, and withdrawing the waste gases from the hood."

ELECTROLYTIC PROCESSES.

Electrolytic Production of Nitrogen Compounds.—J. W. Wood, 826,301, July 17. Application filed Feb. 21, 1902.

The inventor proposes the following peculiar method of pro-

ducing nitrogen compounds from the nitrogen of the air. By means of a nozzle worked with a blower, air bubbles are forced through an electrolyte of dilute nitric acid or sulphuric acid, and serve to aerate the solution thoroughly, and thus charge it with more or less nitrogen. If now electric current is passed through the cell oxygen is liberated at the anode, which is said to unite with the nitrogen and form nitric acid, which is syphoned off. The hydrogen set free at the cathode is also claimed to unite with the nitrogen present, forming hydrate of ammonia. "The nitrogen which˙is thus constantly removed from the liquid is replenished by the nitrogen of the air, as said air passes in the form of bubbles upward through the liquid."

Support for Anodes.—Frank Engelhard, Fred. H. Engelhard and William A. Engelhard, 825,591, July 10. Application filed Sept. 14, 1905.

The object is to provide a device for supporting anodes in electro-plating solutions so that cruder forms of anodes than heretofore can be used, such as the tops or unused portions of old anodes, ingots of any form, etc. These pieces are pierced from side to side and are then slipped on to either a straight or a curved support. The essential feature is that this straight or curved support consists of a strong inner core or rod of brass, which is entirely surrounded in a lead envelope. Details of construction are described and illustrated.

Treatment of Nickel-Copper Matte.—W. McA. Johnson, 825,056, July 3. Application filed Sept. 30, 1903.

Mr. Johnson proposes here a new solution of the old problem of treating nickel-copper matte. (See, for instance, the processes of D. H. Browne, our Vol. I., p. 381, and N. V. Hybinette, our Vol. IV., p. 33.) The copper-nickel matte may contain 39 per cent of nickel, 39 per cent of copper, 1 per cent each of iron and cobalt, and 20 per cent of sulfur, together with small proportions of platinum and palladium. This matte is crushed, ground to 80-mesh, placed in leaching vats, provided with agitators, and subjected therein to the action of a heated solution of hydrochloric acid of about 10 per cent concentration, at a temperature approximating 100° C. Means are provided for conducting away the sulfureted hydrogen formed by reaction of the acid upon the sulfids of the matte. The solution thus obtained contains nickel, cobalt and iron as chlorides. It is freed from cobalt and iron by treatment with chlorine and sodium carbonate or by the action of hypochlorites, according to known methods. The cobalt is recovered from the precipitate as oxide (Co_2O_4), and may be sold as such. The residual nickel-chloride solution is concentrated, if necessary, brought to neutrality or to a faintly acid reaction, and electrolyzed with insoluble anodes, the solution being maintained during the electrolysis at a temperature of about 65° C. Nickel is thereby precipitated in reguline form. and the evolved chlorine is utilized for the manufacture of bleaching powder or otherwise. The residue from the acid treatment may contain 75 per cent of copper, 5 per cent of nickel, and some 20 per cent of sulfur. This residue is subjected to an oxidizing fusion in a reverberatory furnace, and thereby converted into blister copper, the nickel being slagged off and retreated in a small blast furnace. The blister copper is poured hot into refining-furnaces, and therein brought to "set" copper, carrying 0.3 to 1 per cent of nickel. This is then cast into anodes and electrolytically refined to produce cathode copper of 99.94 per cent purity, the precious metal slimes being treated in any approved manner. The nickel salts accumulating in the acid electrolyte are periodically separated therefrom and partially purified from copper by repeated fractional crystallization, the last traces of copper being separated by electrolysis and the nickel salts crystallized for platers' use.

Electrolytic Production of Metals of the Earthy Alkalies.—C. Suter and B. Redlich, 813,532, Feb. 27, 1906. Application filed Jan. 2, 1904.

"A process for the electrolytic production of metals of the earthy alkalies, consisting in melting the salt, effecting the

electrolysis thereof, and causing the slow movement of the cathode from the surface of the electrolyte during the electrolysis." The metal produced at the cathode (for instance, calcium) is thus continually removed from the melt, it becomes gradually solid, and then assumes in its turn the function of the cathode. At the same time the solidified metal becomes covered by adhesion with a thin coating of the electrolyte, whereby it is protected from all oxidation by the air. (This is the method by which calcium is now being made on a commercial scale in Germany. See our Vol. II., p. 276.)

Extracting Gold, Silver, Etc.—J. A. Comer, 813,620, Feb. 27, 1906. Application filed Jan. 2, 1904.

The crushed ore is subjected to a solution of potassium cyanide. The solution is drawn off or decanted from the pulp. The sedimentary deposits suspended in the solution are then separated from it by impregnating the solution with compressed air, rendering the sediments fluffy and flocculent and filtering. The precious metals are then deposited from the solution by electrolysis.

Extracting Metals from Ores.—A. Lénárt, Jr., 826,435. July 17, 1906. Application filed Aug. 18, 1905.

The object is to extract metals from ores by means of the acid radicals evolved in the electrolysis of alkali salts, particularly chlorides. The cathodes are surrounded by diaphragms, to permit removal of the cathodic product of electrolysis. Cathodes and anodes are arranged in direct proximity to each other, so as to reduce the resistance. To make this possible, the ore is not made the anode, and the chamber in which the electrolysis is carried out is arranged beneath the ore chamber in which extraction takes place, and is separated therefrom by a permeable floor. It is stated to be of importance that the "flowing electrolyte should first pass over the anodes and only afterwards over the diaphragms containing the cathodes, so that the electrolyte is already so far saturated with the acid anions before reaching the diaphragms that the lye diffused through the latter cannot precipitate the metals as hydrate or oxychloride from the electrolyte, which during the preceding circulation through the extraction apparatus has absorbed metallic salts."

Manufacture of Sulphuric Acid.—W. McA. Johnson, 825,057. July 3. Application filed Oct. 10, 1903.

Sulfur dioxide, or sulphurous acid, is oxidized in an electrolytic tank provided with anodes of lead, the electrodes being arranged according to the series system—that is to say, in such manner that the intermediate or unconnected plates act as bipolar electrodes. Under these conditions the sulphur dioxide dissolved in the bath is oxidized at the anodes to sulfuric acid. For the cathode surfaces it is desirable to use a metal having a low cathodic potential, whereby the cathodic separation of sulfur is reduced to a minimum. For this purpose copper electro-deposited upon the cathode surfaces is preferred. In practice the procedure is as follows: Sulfuric-acid solution of about 20 per cent concentration is saturated with sulfur dioxide by means of a coke-tower or other suitable device. This solution is then caused to flow through the electrolytic cell and past the electrodes therein, whereby the sulfur dioxide is oxidized to sulfuric acid, and a solution is obtained which may approach 30 per cent concentration. A certain proportion of this acid is withdrawn and concentrated or utilized in any desired manner, while the remaining portion, diluted with water to 20 per cent and again saturated with sulfur dioxide, is returned to the electrolytic cell, this cycle of operation being repeated indefinitely.

Cell for Sodium Chloride Electrolysis.—G. A. Gabriel, 822,-109, May 29, 1906. Application filed Feb. 26, 1906. (Assigned to Bleach & Caustic Process Co.)

The cell is a modification of the Hargreaves type. The special feature is the means for leading the liquid, which percolates through the porous diaphragm, away from the diaphragm, thus obviating the danger of back percolation. The

method is indicated in the left-hand diagram of Fig. 4, where 13 is the porous diaphragm, and 11 a forminous screen forming the cathode proper. A series of downwardly-inclined wires 15 is provided, the upper ends of which are twisted around the strands forming the screen 11. These wires 15 form conduits, which, owing to the adhesion of the liquid percolating through the diaphragm, lead the liquid away from the diagram and the screen itself, so that the liquid drops from the ends of the wires or is conducted to the "drain-plate" 16. The holes 22 permit the liquid to flow over both sides of the plate, and therefore increase its draining capacity. The liquid collects on the bottom of the cathode chamber, from where it is removed. For making the diaphragm the apparatus, shown in the right-hand diagram, is used. A rectangular frame 30 has attached to it on its four sides a series of coiled springs 31 with hooks 32, which pass through and engage the diaphragm 13 to be
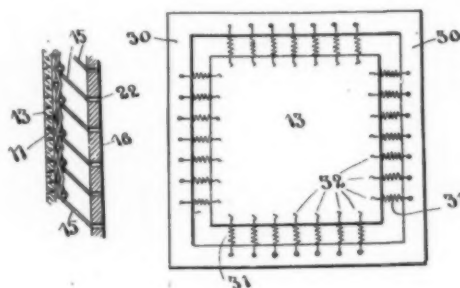

FIG. 4.—DETAILS OF CONSTRUCTION OF SODIUM CHLORIDE
ELECTROLYTIC CELL.

treated. The fabric when thus stretched in the frame is moistened and allowed to dry under the tension of the springs 31. When once dried, the fabric is again dampened, and is then covered by a coating of cement and water, or clay, and silicate of soda, etc.; this mixture is rubbed thoroughly into the interstices of the fabric, and the surfaces made smooth and even on both sides. The fabric and cementitious material is then allowed to set and harden to the consistency in which it is to be used, after which the screen is removed from the frame and placed in a mold to give it the requisite shape.

Bleaching Liquor.—F. L. Bartelt, 813,688, Feb. 27, 1906. Application filed Oct. 7, 1904.

The object is to provide a convenient and automatic arrangement for the production of bleaching liquor in small steam laundries. On a raised platform a belt-driven dynamo (or a converter, etc.) is placed together with the "solution tank." This is provided with an indicator by which the height of the solution in the tank may be seen, and by which the filling of the tank may be regulated. From the bottom of this tank connection is made through several independent pipes with the "electrolyzer," which is placed at a lower level. The bleaching liquor produced flows off into the "liquor tank," which is at still lower level. In the solution tank is provided a float, adapted under certain circumstances to close one or more circuits. Thus it may close a bell circuit and an electrolyzing circuit. When the level of the solution in the tank descends below a certain point, the float closes both these circuits, so that the bell is rung to call the attention of the attendant, and at the same time the switch is operated to cut off the current to the electrolyzer, as otherwise the current would produce a liquor detrimental to the clothes.

Electrode for Production of Bleaching Liquors.—R. Kother, 806,413, Dec. 5, 1905. Application filed Sept. 7, 1905.

A bipolar electrode is constructed as follows: A thin platinum-wire netting or platinum foil is placed across a glass or hard-rubber plate, so as to cover entirely the anode side. But the platinum is also folded round the edge of the glass plate

and a graphite plate which forms the cathode is pressed against the turned edges of the platinum so as to make good contact.

Treating Organic Substances in Presence of Vanadium Compounds.—E. Oppermann, 823,435. June 12, 1906. Application filed Aug. 31, 1904. (Assigned to Meister, Lucius & Bruening.)

The inventor makes use of the great oxidizing power of the higher oxides of vanadium and the energetic reducing power of the lower oxides of vanadium. Vanadium compounds in an electrolytic bath are excellent depolarizers, both at the anode and cathode. The rapidity of carrying oxygen and nitrogen to the "acceptors" being very great (indigo is reduced instantaneously, and one part of vanadium salt oxidizes 1,000 parts of aniline to aniline-black in presence of oxidizing agents), only small quantities of vanadium salt need be used for reducing or oxidizing by electrolysis. The various oxides of vanadium produced in the electrolytic bath have a more or less strong reducing or oxidizing power. As an example, the oxidation of anthracene may be described: Into a vessel lined with lead, serving as anode, is introduced sulphuric acid of 20 per cent strength, with a quantity of vanadic acid dissolved in it, for instance, 1 per cent. The cathode may be of lead. The anthracene to be oxidized is introduced into the vessel as a fine powder while actively stirring, and the mixture is heated to about 80° C. With a current density of 300 amps. per square meter the voltage is 1.6. After a short time anthraquinone may be proved. If somewhat more ampere-hours than the theoretical figure have passed the cell, the anthraquinone need only be filtered, and the electrolyte may be used for another operation. Other examples dealt with in the specifications are the oxidation of aniline to quinone (hydroquinone), the reduction of azobenzene, and the reduction of azoxybenzene.

Electrolytic Production of Hydrogen and Oxygen.—L. van Scoyoc, 813,844, Feb. 27, 1906. Application filed May 15, 1905.

Details of construction of an electrolytic apparatus for the production of hydrogen and oxygen, in which the operation is rendered continuous by the use of automatic float-valves. The level of the acidulated water in the electrolyzer is maintained constant by means of a float-valve in the supply pipe. The two electrodes are placed in two compartments which, of course, are open at the bottom. Each compartment is divided into a lower and an upper chamber, connection between the two being made by automatic float-valves. When the pressure of the gases generated in the lower chambers becomes great enough to lower the level of the water, the valve is opened and the gases pass into the other chamber and gas-bags.

Electrolytically Preparing Metals for Lithographic Purposes.—O. C. Strecker, 810,889, Jan. 23, 1906. Application filed April 19, 1900.

Metal plates for printing purposes are generally prepared by coating them with a layer of hygroscopic materials directly or by brushing, powdering, etc., in a mechanical way or by coating the plates with materials capable of forming a hygroscopic layer by chemical reaction with the metal of the plate. The present inventor employs an electrolytic process for the same purpose, the plate being used as electrode before or after transferring the drawing. The procedure is as follows: A zinc plate is ground by means of a pad of steel turnings and pumice powder, covered with soft leather. The metal is then rinsed with water and dried. On this plate a lithographic design or transfer, the negative of the intended print, is fixed. After this the plate is gummed with a solution of gum-arabic of medium strength and dried. After being perfectly dry, the greasy substance is washed out by lithophine (a solution chiefly consisting of asphaltum in benzene or spirits of turpentine). The excess is wiped off and the plate dried. Then the gum-arabic is washed off the plate by water, and the negative is left on the plate as a layer of asphaltum. The negative is properly inked

by means of the roller, and then the plate is cleaned from spots. Alterations and corrections are made with lithographic ink in the well-known manner, and the plate is now ready for electrolytic treatment. It is used as anode in a 2.5 per cent sodium fluoride solution, with a current density of 0.1 to 0.5 amp. per square foot, for 2 to 5 minutes. The reaction is $3Zn + 6NaF + 6H_2O = 3ZnF_2 + 6NaOH + 6H$. The alkali formed has to be neutralized before using the electrolyte again. The whole plate may either be suspended vertically in the solution ("dipping method"), or it may be placed horizontally, solution poured on it and a cotton-covered metal plate, forming the cathode, passed over it ("sponging method"). For further details reference must be had to the specification.

Electroplating.—S. H. Thurston, 822,873, June 5, 1906. Application filed April 1, 1905.

The object is to make the electrolytically deposited metal stick to the surface on which it is plated. For this purpose this surface is first treated mechanically. The surface receives an "amorphous, inherent, adherent, coherent and permanent coating" of the desired metal by heating particles of this metal against and into the surface. For this purpose the inventor uses a beating machine, in which "beating rods" of wire (of the desired metal) are suspended from a revolving axle and are beaten against the surface. The rough coating thereby produced on the surface serves as an "anchorage" for the electrolytic deposit which is applied afterwards. Among other applications the inventor states that this method enables one to plate on aluminium.

Sterilizing Milk.—R. C. Turner, 806,600 and 806,601, Dec. 5, 1905. Applications filed Jan. 25, 1904, and March 16, 1904.

The milk or other liquid, to be sterilized, is passed through a series of vessels, flowing in a stream from one vessel to the next. Simultaneously an electric current is passed from the bottom of one vessel upward through the body of milk, then downward through the stream, falling from this vessel to the next, etc. For this purpose an electrode is placed on the bottom of each vessel. The electrodes may be connected to the electricity supply in different ways. For instance, the electrodes in the first, third, fifth vessels may be connected to the positive pole, those in the second, fourth, sixth vessels to the negative pole.

Purification of Water.—L. Dion, 820,482, May 15. Application filed May 24, 1904.

Mechanical details of construction of a cell in which water is to be purified by being passed between series of electrodes.

BATTERIES.

Converting Spongy Lead Into Sulphate.—C. J. Reed, July 17. Application filed May 7, 1903. (Assigned to Security Investment Co.)

To convert a spongy lead plate into lead sulphate without using an external current, the inventor forms a short-circuited couple, consisting of the lead plate and another plate provided with a coating of platinum black, in a dilute solution of sulphuric acid.

Storage Battery Plate.—A. E. Knight, 826,173, July 17. Application filed Nov. 8, 1905.

The electrode is composed of two grids, each of which consists of a thin sheet of lead with punctures distributed over it and with ribs attached to one face, which form compartments into which the active material is firmly packed. The grids are secured together with the perforated sheets on the outside, and the active material in the chambers of one grid is separated from the active material in the chambers of the other grid by a thin film of lead foil, which is firmly clamped between the grids to make a good electrical contact therewith, whereby the conductivity of the electrode is increased. The film of lead foil is provided with very fine perforations.

Separator.—L. W. Horton, 825,837, July 10. Application filed March 17, 1904.

The separator comprises two parts arranged face to face,

the adjacent faces being provided with longitudinal and transverse grooves, so as to provide intercommunicating acid wells within the bounding surfaces of the separator. The object is to have as large a portion of the active material as possible in contact with the electrolyte.

Battery.—C. W. Harper, 825,882, July 10. Application filed April 8, 1904.

Details of construction of a pocket battery to be used in connection with a portable telephone system for deaf persons.

Storage Battery.—G. A. Ford, 824,348, June 26, 1906. Application filed July 20, 1904.

The battery is built up of trays or pans in a substantially horizontal position, above one another. The trays are made of gutta-percha, and have a perforated bottom. The perforations are plugged with a porous material. Perfect circulation of the electrolyte is obtained by means of special openings. Within each pan an electrode is provided of peculiar construction. There is a central web with vanes between it, and the edges of the vanes lie at an angle to the plane of the plate. An auxiliary electrode-plate has a web coextensive with the other web, and is provided with fins projecting into the spaces between the vanes.

Storage Battery.—H. H. Porter, 813,52, Feb. 27, 1906. Application filed Sept. 21, 1903.

The patent refers to the construction of a separator between the plates. It consists of a thin layer of wood with parallel rectangular vertical grooves arranged alternately on its opposite sides, and, between them, relatively wide flat spaces, each such space being directly opposite a groove on the other side.

Storage Battery Plate.—William L. Silvey, 824,828, July 3. Application filed March 5, 1906.

The construction is indicated by the first claim: "In a secondary battery a perforated battery-plate, consisting of two lead plates having holes in them larger on the inside, perforated metal cups fitted into the holes in the perforated lead plates, the perforated lead plates fastened together forming perforated cavities for active material, and connecting ears on the plate for connecting it to an external circuit."

Storage Battery.—A. Mueller, 813,730, Feb. 27, 1906. Application filed July 30, 1904.

The battery electrode consists of an envelope, comprising two perforated covers or two sheets of metal gauze, within which are contained cakes of active material. At their margins these covers are held together by U-shaped reinforcing strips. At suitable intervals throughout the covers there are loops through which pins are inserted, which firmly bind the two covers together and stiffen the plate.

MISCELLANEOUS.

Ozone Ventilator.—F. de Mare, 820,656, May 15, 1906. Application filed Sept. 23, 1904.

The patent relates to a centrifugal fan, and the special feature of the device is that it sucks in, ozonizes and delivers the air in a single operation and in the same apparatus. A porcelain or glass casing of the usual form of the shell of centrifugal fans is divided into two identical halves by means of a vertical glass plate in the center, perpendicular to the axle of the machine. The axle carries two metal sockets insulated from each other, one in one section, the other in the other section of the machine. Each metal socket carries a series of aluminium wings which are electrically connected to a source of high-tension currents. Briefly, the device is an electrifier and pump for the gas it electrifies.

Locating Metals Beneath the Earth's Surface.—E. R. Wolcott, 822,175, May 29. Application filed Sept. 5, 1905.

"This invention consists of a process of locating and also extracting metals beneath the earth's surface. The process consists in placing electrodes in the ground at a suitable distance from each other, moistening the earth between the electrodes, if necessary saturating the earth with acids adapted to dissolve the metals contained in the earth, and finally passing a current of electricity from the anode to the cathode. In this event the dissolved metals will be carried along with the electrical current and deposited at or upon the cathode." By changing the position of the electrodes, the location of the metal or ore in the earth is determined. The electrodes are enclosed in porous cups filled with electrolyte; the anode consists of carbon, the cathode of platinum. Nothing is said how the scheme works in the field and not on paper.

RECENT METALLURGICAL PATENTS.

LEAD AND SILVER.

Desilverizing Lead Bullion.—In the old Parkes process the procedure is generally as follows: After the bullion has been freed of its copper, antimony and arsenic in a "softening" furnace, metallic zinc is added and stirred therein, to cause the contained gold and silver, for which it has an affinity, to rise to the surface in the form of a scum containing more or less lead. This scum or zinc skin is skimmed off from the liquid lead by means of perforated ladles, and undergoes further treatment by retorting of the zinc and cupelling to eliminate the lead, and thus obtain silver and gold. According to A. Raht (826,114, July 17) a small quantity of sal-ammoniac, when added shortly after the zinc has been introduced and thoroughly stirred with the molten metals, effects a more complete separation of the lead from the scum than is possible by mechanical means alone. This supplementary step in the smelting process not only thus reduces the bulk of scum to be operated upon, but likewise causes it to be in a "drier" state; i. e., containing less lead, and consequently more easily and cheaply treated in the following refining process. The cost is said to be slight, 1 pound of sal-ammoniac being required per 10 tons of bullion.

NICKEL.

Mond Process.—For the production of nickel from "nickel carbonyl" (our Vol. II., p. 291), the vapor of the latter may be passed through nickel shot or pellets kept at a temperature of about 200° C. The nickel deposits from the vapor on the shot, which gradually increase in size. Carl Langer, of the Mond Nickel Co. (825,844, July 10), points out that for the proper working of the apparatus it is essential that the pellets should be as nearly as possible at the same temperature at all parts of the cylinder containing them. If this is not the case, the nickel carbonyl vapor may pass among the colder pellets without being decomposed, and nickel will be deposited on the hot surface of the cylinder. Moreover, the hotter pellets may decompose the carbon monoxide, causing a deposit of carbon on the nickel. There is also the danger that undecomposed vapor may arrive at the outlets, and, becoming heated in the neighborhood thereof, blocks them. To control the temperature better, Langer heats the cylinder by a number of gas-flames, evenly distributed around the outside of the cylinder. The latter is provided with projections or ribs, to form, with the outer casing, chambers to contain the gas-jets. The part of the cylinder where the escape of gases occurs is formed as an annular chamber through which a cooling medium is circulated, the gas-escape pipe passing through this chamber.

Treating Nickel and Copper Matte.—A patent of J. A. Gilman (826,099, July 17) relates to a modification of his and Everett's process, whereby the nickel and copper matte while in the molten state is finely sub-divided into particles by impingement against a mechanical surface, like a cooled screen, the stream of particles being then subjected to the action of the oxygen of the air, so as to remove the sulphur from the particles. In the new process the descending particles are met by a lateral air-blast, and thereby deflected, oxidized and desulphurized. The object is to decrease the height from which the

particles must be projected and allowed. An account of a new process of W. McA. Johnson for separating copper and nickel may be found in the Aanalysis of Current Electrochemical Patents in this issue.

COPPER.

Copper Castings.—J. A. Yunck (825,100, July 3) proposes to deoxidize copper by means of nascent sodium vapor, to prepare it for the casting of sound and flawless ingots. He prepares a mixture of 30 parts of powdered carbon, 5 of coarser or granular carbon, 5 of carbonate of lime, and 60 of sodium carbonate, thoroughly mixes in water until the sodium carbonate dissolves, then evaporates to dryness and crushes the residue. This mixture is mixed with copper in the proportion of 1 to 19 parts, and is fused in a crucible sealed except for a small vent. When the temperature is raised to or above the melting point of copper, the sodium carbonate and carbon combine, forming sodium vapor and carbonic acid. The sodium combines with the oxygen of the oxides in the metal, and sodium oxide raises to the surface of the bath and forms a crust. The carbonate of lime in the mixture is intended to act as a retarder, to give the sodium vapor time to do its work.

TIN.

Detinning Tin Scrap.—On account of the comparative scarcity of tin ore, and of the immense quantities of tin used for tinning sheet iron, the electrolytic detinning of tin scrap has proven quite economical in recent years. In this development the firm of Theodor Goldschmidt, in Germany, has performed decisive pioneer work. It is noteworthy that in spite of their flourishing electrolytic works this firm has experimented in recent years on a purely chemical process of detinning the scrap, as is shown by a patent recently granted (822,115, May 29) to one partner of the firm, Dr. Karl Goldschmidt, and to Mr. Josef Weber. The process consists in subjecting tin scrap in a compressed state to the action of chlorine in combination with an "inert anhydrous fluid," in which the chlorine gas is dissolved. By the term "inert anhydrous fluid" an anhydrous fluid is meant which does not attack the scrap or chlorine or the products thereof during the process of detinning. Tetrachloride of tin is stated to be suitable for this purpose.

FIG. I.—DETINNING PLANT.

The process seems to be extremely simple in operation, only few precautions being necessary. It is important to avoid any great increase of the temperature during the process. Further, since the tin scrap is used in firmly compressed bundles, care must be taken that the chlorine enters even into the narrowest and most firmly closed spaces. This can be accomplished by alterning the pressure in the closed vessel during the process, and it is of advantage to guide the process in such a manner that the pressure in the vessel is so augmented that at the end of the operation the over-pressure is, say, one atmosphere above the ordinary pressure.

The bundles of tin scrap may conveniently be 40 to 60 centimeters in width and length, and 8 to 14 centimeters in height, so that they weigh from 50 to 60 kg. The detinned iron bundles may then be directly used in steel works. The iron waste is obtained perfectly detinned with a fine smooth gray surface.

The apparatus is shown in Fig. 1; *a* is the iron receptacle for the compressed tin scrap bundles *s*. In order to facilitate the filling of the vessel the packets are brought in baskets *d*,

and these baskets, filled with the bundles of compressed tin scrap, are then introduced into the vessel. Fluid chloride of tin, in which chlorine gas is dissolved, is then admitted through the pipe *t*. The chlorine gas enters the receptacle *b* through the pipe *t'*, and the chloride of tin comes into the receptacle *b* through the pipes *n* and *c*. The fluid chloride of tin comes in contact with the surfaces of the tin scrap, and the chlorine dissolved in the fluid chloride of tin acts upon the tin of the tin scrap, and chloride of tin is formed. The fluid chloride of tin flows down and new fluid chloride of tin, in which chlorine gas is dissolved, is led from above into the receptacle. The fluid chloride of tin is gathered at the bottom of the receptacle and is lifted by a pump *p* through the tubes *c* and *c'* into the receptacle *b* into which, by the tube *t'*, chlorine gas is led, so that chlorine is dissolved in the fluid chloride of tin. This fluid chloride of tin, in which chlorine gas is dissolved, is brought anew through the tube *t* from above into the receptacle *a*, and the reaction of the chlorine begins again.

If the operation is as simple as described in the specification the process looks very promising, and is presumably cheaper than electrolytic detinning. It might represent an important outlet for electrolytic chlorine, in view of the fact that our electrolytic caustic and chlorine plants have troubles in working their chlorine into profitable products. While electrolytic detinning yields metallic iron and metallic tin as end products (the value of both metals being in general about equal), the final products of the new process are iron and tin tetrachloride. There is a good market for the latter product, which by treatment with ammonium chloride may be easily changed to pink salt, which is extensively used in cotton printing and silk dyeing. The tin tetrachloride could also be treated with water and afterwards with metallic zinc, resulting in the precipitation of metallic tin and formation of zinc chloride. See our Vol. II., p. 339.

LIXIVIATION PROCESS.

Sulphatizing Ores.—E. Enke (826,925, July 24) patents the following process for the sulfatization of ores: The ores, their by-products and the like, are mixed with such quantities of acid alkali sulfates or ferrous sulfate, or mixtures of both, that the alkalies which, at the fusion, form sulfates, receive the required quantity of sulphuric acid from the added compositions of sulphuric acid. In preference sodium bisulphate is mixed with the ferrous sulphate. It is essential that the air be excluded during the melting, and that the temperature be raised to such a degree that the ferrous sulphate, which is formed at the beginning, is separated into oxide of the iron and sulphuric acid, so that the latter can combine as soon as it is formed with any alkali present in the charge and adapted to form a sulphate, so that this sulphuric acid is not lost and a considerable saving in bisulphate to be added is obtained. The temperature at which the other metallic sulphates decompose is, however, not reached, so that upon lixivation a solution is obtained which is free from iron and contains as little metallic sulphate as possible. Ferrous sulphate is employed particularly for the purpose of enriching the charge to be lixiviated so much with iron that the lixiviated residue forms a marketable product.

TREATING MATTE.

Removal of Iron from Matte.—To treat matte for the object of removing iron, the matte may be blown, in order to cause the iron to enter the slag. But in this method practically all the sulphur contained in the matte is burned to sulphur dioxide, which escapes into the air; further, the silica is taken from the lining of the Bessemer converter, whereby the lining is destroyed. Of course, the matte may first be roasted and then smelted with sand in a shaft-furnace, in order to slag the iron oxidized by the roasting. Besides being very costly, due to the large fuel consumption, the roasted product is in a pulverulent condition and requires briquetting before smelting. Moreover, the roasting must proceed with care, in order to leave sufficient sulphur in the mass for the subsequent smelting.
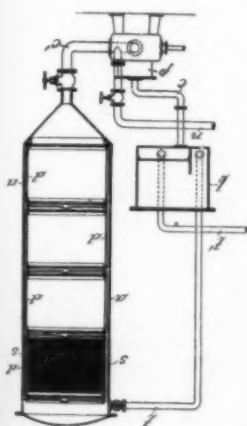
J. Savelsberg, of Papenburg, Germany (825,983, July 17), endeavors to avoid these objections in the following manner: The matte is finely divided and mixed with the necessary percentage of slag-forming ingredients, such as sand and lime, requisite for the subsequent slagging of the iron. This mixture is then brought onto a bed of glowing material, as coal, coke, ore, etc., in a suitable apparatus, and a blast is passed through the mixture from below. Due to the heat of the reaction, the mass gets red-hot without completely melting, so that the mass sinters together. The blowing is completed when the red-hot mass cools down, which it does very rapidly. During the blowing, practically all the sulphur, combined with the iron, has combined with nickel, copper, etc., while practically all the iron has been oxidized by the air-blast. The mixture after cooling is broken into pieces of suitable size for subsequent smelting in a shaft-furnace, which will yield a matte practically free from iron, yet containing substantially all the sulphur of the previous matte, while the slag is quite fluid, and contains only a small percentage of nickel, copper, etc., and nearly all the iron in the matte previously employed. A matte containing 50 per cent nickel, 14 per cent sulphur and 36 per cent iron, contained after such a treatment (i. e., blasting and subsequent smelting) 77 per cent nickel, 20 to 21 per cent sulphur, and ½ per cent iron. Thus it will be seen that hardly any sulphur was lost, since the proportion of nickel to sulphur in the resulting matte is the same as in the matte treated.

ROASTING FURNACES.

McDougall Furnace with Helical Hearth.—The well-known McDougall furnace has a number of successive hearths, from one to the other, of which the material is dropped in the process of roasting. This results in the production of much flue-dust. R. L. Lloyd and P. Thill (12,511, July 24) propose to overcome this objection by providing the McDougall funace with a helical continuous hearth (like Fig. 1, in our Vol. III., p. 120), making as many turns of the helix as to get sufficient surface for the proper roasting of the material passing through the furnace. So far, the arrangement is undoubtedly sound. But now arises the question how to arrange the stirrer arms between the turns of the helix, for it is manifest that the operation of arms in the helical spaces by continuous or rotary motion is impossible. The inventors, therefore, provide for a rotary reciprocation of the arms and arrange for their following the pitch of the helix in their downward movement (where the arms are angularly placed 90° apart) for a little more than half a revolution. At the end of their stroke they are first raised vertically a sufficient distance to clear the body of ore on the hearth, and are then rotated backward, being depressed at the beginning of the stroke until they again engage the ore on the hearth. But if we consider that the central shaft and stirrer arms of a commercial furnace weigh several tons, it is clear that a very considerable amount of power would be required to operate the stirrer mechanism. The inventors endeavor to overcome this difficulty in a very interesting manner by the expedient of counterbalancing the rabble shaft and arms so that as these are lowered the counterweight is raised. They carry out this idea by supporting the mechanism upon hydraulic cylinders and connecting the latter by pipes of sufficient capacity. Thus as one sinks the other rises, and the amount of power required to operate the furnace will be, as far as the lifting is concerned, greatly reduced. The design is certainly interesting, but it is questionable whether the advantages of the continuous helical hearth are not outweighed by the complications of the stirring mechanism. This question must be decided in practice.

Cooling of Shaft and Raffles.—Another patent (825,326, July 10) has been granted to Frank Klepetko for a method of cooling the rabble-shaft and rabble-arms of a McDougall furnace. Shaft and arms are hollow, and within the shaft is located a feed-pipe, through which a limited quantity of water is injected into one compartment of the shaft, the latter being divided into a series of compartments. By evaporation the steam fills the compartment into which the water is introduced, passes then into the next rabble-arms and back through special conduits provided within the arms to the next compartment of the shaft, etc. The cooling effect is due to the superheating of the steam in its passage through shaft and arms.

Rake Construction.—A patent of G. S. Crouse (825,519, July 10) relates to mechanical details of rake construction of McDougall furnaces, the object being to provide improved means for attaching the rakes or blades to the arms, consisting essentially of improved split boxes or frames held together on the arms by the rakes or blades.

Tilting Roasting Furnace.—In the old form of "Edwards tilting" furnace—which is a reverberatory furnace of elongated form with revolving rabbles—the capacity is limited, because the width of the furnace is limited by the circle swept by the rabble-arms. Long arms are impracticable for mechanical reasons. T. Edwards (825,446, July 10) now provides several rows of rabbles in the ore chamber with intersections of each rabble-hearth area by other rabble-hearth areas. This permits to make the furnace of any width and capacity, although short-armed rabbles are employed. The whole furnace consists of two longitudinal chambers, one above the other, with a connection between them. The upper chamber is the ore chamber, as already described, while the lower chamber is sub-divided by a longitudinal partition, with compartments for the entrance and exit of highly heated gases and furnace products. Dampers control the outlets of the chambers.

REGENERATIVE GAS FURNACE.

Heat Regulating Device.—Archibald Johnston (824,723, July 3), the general superintendent of the Bethlehem Steel Co., improves the regulation of heat in a regenerative furnace as follows: Fig. 2 shows a transverse vertical section with the furnace chamber A, along each side of which extends the longitudinal regenerating gas chamber B and the regenerating air chamber C, both being connected to sources of gas and air respectively, and the connections being controlled by the usual reversing valves in the well-known manner. The new feature is that the "heating fluid" (produced by the combustion of gas and air) may be caused to change its path through the furnace chamber. For this purpose a plurality of entrance ports are provided for the gas as well as for the air. The illustration shows the two entrance ports 4 and 5 from the air chamber to the furnace chamber.
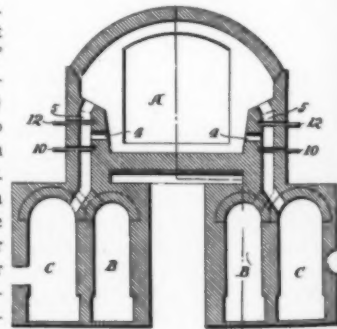

FIG. 2.—HEAT REGULATION.

In the same way upper and lower entrance ports lead from the gas chamber to the furnace chamber. If valve 10 is open and 12 closed, the air is forced through 4; if 12 is opened the air will pass upwards through 5. The air and gas supply is regulated in quantity by the lower valves (like 10), while by means of the upper valves (like 12) the air is directed to the upper or lower outlets, or both, as desired. If the material to be heated is placed on the floor of the furnace chamber, and if the gas and air are passed through the lower ports (like 4), the heating fluid will be in direct contact with the material to be heated. If the gas and air are introduced through the upper ports (like 5), the heating fluid will not come into direct contact with the material to be heated, and the latter will be heated by radiation. In this case

oxidation is prevented and the formation of scale on the surface of the material is avoided.

IRON AND STEEL.

Reverberatory Steel Furnace.—V. Defays (825,522, July 10) patents details of construction of reverberatory furnaces for the production of steel, the object being the possibility of using poor gases, such as blast-furnace gas, and to obtain perfect control of the flame. The chief feature is the arrangement of the gas flues and air flues. At each extremity of the furnace a gas-flue opens through a suitable aperture into a combustion chamber between the apertures of two air flues, in such a manner that a sheet or layer of gas is arranged between the sheets of air, the axes of these three flues being preferably convergent, so that the jet of gas may be attacked, for example, from left to right at its upper part by one of the air jets, and from right to left at the lower part by another jet, so that it assumes a gyratory movement, insuring intimate mixture between the combustible and the carrier for oxygen. The combustion chamber constitutes a removable part independent of the metal chamber of the furnace and of each of the air and gas supply flues, which are also removable.

Attachment for Blast Furnaces.—C. Abercrombie (826,819, July 24) patents an attachment for removing the furnace-stock, carried away with the gases of blast furnaces, so that these gases are freed from any fine dust; the free discharge of the fine ore from the mouth of the furnace is taken care of so as to prevent a discharge of the fine ore at the mouth of the furnace in clouds. For this purpose the inventor provides several "primary" conducting pipes leading to the "furnace-stock catcher," which is in communication with a "furnace-stock collector." "Secondary pipes" communicate between the collector and the furnace; they serve only in cases of violent explosions or slips. The collector and catcher contain a body of water of the same level, and a gas-conducting pipe communicates with the catcher near its top. The water in the catcher and collector removes the coke, ore and flue dust contained in the gases, and the matter which is caught in the furnace-stock catcher flows by gravity into the collector, from where it can be easily removed. A spray-pipe in the catcher near the top insures quick removal of the fine dust from the gases.

Iron-Ore Reduction.—G. L. Fogler (826,557, July 24) tries to reduce iron from its ore in a stack furnace under diminished pressure by the action of a highly-heated reducing gas made in a gas producer. Each twyer is supplied with superheated producer gases by a pipe projecting into its twyer end, and leaving between the gas pipe and twyer an annular space in communication with the atmospheric air of such size as to admit under the siphoning action of the gas flow, and against the resistance offered by the charge a proportion of air insufficient to effect complete combustion. This is regulated so as to produce the greatest increase in the temperature of the gases without impairing their action in reducing the ores. The incompletely-burning gases reduce the ore, rapidly leave the reduction zone, expand and flow quickly upwards, both because of the greater porosity of the upper charge, and because of the partial vacuum which is maintained at the upper end of the furnace.

MISCELLANEOUS.

T. Evans (826,715, July 24) patents mechanical details of construction of an air-tight blast-furnace twyer, with automatic valve.

J. Levey (825,064, July 3) patents details of a valve for gas generators, the special object being to control the flow of hot or otherwise destructive vapors.

M. Hoopes (825,537, July 10) patents the construction of an oil furnace for heating metal blanks, the latter being automatically conducted through the furnace and thereby heated.

W. L. Painter (825,003, July 3) patents construction of a heater with twyers and a forced air blast, specially adapted for use in heating tires.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

FIXATION OF ATMOSPHERIC NITROGEN (CALCIUM CYANAMIDE).

A most interesting lecture on this subject was presented at the recent Congress for Applied Chemistry in Rome, by Prof. Dr. ADOLPH FRANK, of Charlottenburg, Germany, who fifty years ago was instrumental in making the Stassfurt potassium salts available for fertilizing purposes, and who during the last ten years has devoted much time and thought to the problem of fixation of atmospheric nitrogen.

Together with Dr. N. Caro, Prof. Frank observed that barium carbide BaC_2, when heated to a high temperature, binds the nitrogen almost quantitatively, according to the equation $BaC_2 + N_2 = Ba(CN)_2$, thus yielding barium cyanide. Their first patents were taken out in 1895. The Siemens & Halske Co. then became interested in this work, and a new company, the "Cyanid" Co., was formed.

The production of cyanide appeared at that time most profitable, in view of the rapid increase of consumption in the cyanide process for gold production. But, with the Boer war and the stopping of gold mining in Transvaal, the price of potassium cyanide went down suddenly, and Frank and Caro were forced to try and cheapen their method. They now took up again older experiments, tending towards the substitution of calcium carbide for barium carbide as starting material in their process.

Here they found, however, that the reaction was different, and that calcium cyanamide $CaCN_2$ was formed in preference to calcium cyanide CaC_2N_2, the former according to the equation $CaC_2 + 2N = CaCN_2 + C$. But the calcium cyanamide which is formed in this way, together with the cyanide, can be made to bind again the second atom of carbon (see last equation), by melting it with alkali salts, so that in this way pure cyanide may be produced.

More important, however, was the establishment of the fact that the raw calcium cyanamide, when heated with water under high pressure, reacted according to the following equation: $CaCN_2 + 3H_2O = CaCO_3 + 2NH_3$. This represented the solution of the old problem of making ammonia from atmospheric nitrogen.

The fact that this simple reaction takes place led the son of the author, Dr. Albert Frank, to the supposition that the raw calcium cyanamide might be used directly as a fertilizer. Practical tests, extending over several years, have shown, indeed, that raw calcium cyanamide with a content of 20 per cent of nitrogen is fully equivalent to ammonium sulphate with the same content of nitrogen.

Of the raw materials required in the process, lime, carbon and the atmospheric nitrogen are everywhere available, but the problem of getting pure nitrogen from the atmosphere had to be solved. The author found that a satisfactory commercial way for getting pure nitrogen on a large scale is by means of

distillation of liquid air (the pure oxygen which is hereby obtained as a by-product is to be used in a later stage of the process, which will be mentioned below).

Of fundamental importance for the commercial success is, however, the supply of sufficient electrical energy at a reasonable price. This is not available now in Germany. When Prof. Angelo Menozzi, in Milan, became interested in the process, the Societa Generale per la Cianamide was formed in Rome, which acquired all the patents for making calcium cyanamide and its derivatives. This company sold the Italian and Austrian patents to the Societa Italiana per la Fabbricazione di Prodotti Azotati, which has already started a large plant in Piano d'Orte. Since in this plant the process proved satisfactory in every respect, it was decided to greatly enlarge the works by developing the large water powers of the Pescara River, which are owned by the mother firm, and also to erect a large factory in Fiume. A number of other plants will be

CALCIUM CYANAMIDE PLANT IN PIANO D'ORTE.

erected in France, Spain, Switzerland and Norway, at places where cheap water power is available.

It is hoped that in Germany cheap energy may be obtained from peat, and that the large electricity works which are operating at full load only during 20 or 25 per cent of the whole time might take up the manufacture of carbide during hours of low load.

Calcium carbide is one of the best energy accumulators. With respect to the binding of nitrogen by means of carbide, the theoretical figures are that an electric horse-power-year produces the quantity of calcium carbide required for binding 772 kilograms of nitrogen (corresponding to about 5,000 kg. of Chili saltpeter). In practice, however, this theoretical figure is not reached by any means, since 1 hp-year suffices only for the binding of 300 or 330 kg. of nitrogen; anyhow, this corresponds already to 2,000 kg. of Chili saltpeter, or 1,600 kg. of ammonium sulphate.

As mentioned above the reaction of calcium cyanamide with steam at high pressure proceeds almost quantitatively. It yields nitrogen in form of ammonia, and the efficiency of this process when operated on a large scale is 96 to 97 per cent.

Ammonia obtained in this way can be oxidized by means of the oxygen (obtained as a by-product from the distillation of liquid air) to nitric acid, and experiments which Dr. Frank and Caro have made lead them to the conclusion that this more roundabout way of making nitric acid from atmospheric air may give better results with respect to power consumption and output than the more direct process of Birkeland and Eyde by electric discharges through air.

Other interesting applications of the process are the preparation of organic compounds, for instance, the production of artificial indigo by Caro's process, already patented in Italy, in which the action of dicyanamide on phenyl-gycine and its derivatives is used. More recently di-cyandiamide and its salts

have been used as an addition to explosives and powder for the purpose of lowering the temperature in the gun barrel. On account of the high content of inert nitrogen (66.6 per cent) in dicyandiamide, it yields strong pressures in the gun-barrel while it produces only a small quantity of heat. This is of importance, since it lengthens the life of the gun-barrel. In some mixture of powder the cooling action of the dicyandiamide makes the fire disappear at the end of the gun-barrel so that we have now powder which is not only smokeless but fireless.

Ludwig Loewe & Co. (large manufacturers of guns and machines in Berlin) found that calcium cyanamide may also be used for hardening steel. Various tests have given such good results with tool steels, ore drills, pieces of machinery and armor plates, that this new hardening material has been placed on the market under the name of "ferrodur," and has already found an extended application in fine mechanics' workshops as well as in the machine shop.

COPPER.

Mining, Milling and Smelting in the Cifton-Morenci-Metcalf District.—A serial in the *Engineering, and Mining Journal*, by D. E. Woodbridge, dealing with the general conditions of mining and metallurgy in Arizona and Sonora, contains in the July 21 issue of that paper some notes on the milling and smelting methods used at the properties of the Detroit Copper Co. The greater part of the ore mined requires concentration, the concentrating ore varying from 3 to 5 per cent of copper, and being mostly of the lower grade. The mill concentrates 7.5 tons into one, the product containing 18 per cent of copper, 23 per cent silica and alumina, and 30 per cent sulphur, the tailing loss amounting to 0.8 to 0.9 per cent of copper. The ore is crushed so that 50 per cent or more will pass through a 100-mesh, and 35 to 40 per cent of those fines through a 200-mesh screen. The consumption of water is about 225 gallons per ton of ore treated. The completed mill is expected to treat at least 1,400 tons per day. The mill is operated by Crossley-type producer gas engines, 600 hp. being installed for the full mill. The smelter contains in addition to smaller furnaces one large one of the usual water-jacket type, the dimensions being 264 x 48 inches; the distance from the charging floor to the tuyeres is 14 feet 10 inches. The daily charge passed through this furnace amounts to 400 tons, mainly of sulphide concentrates. A reverberatory furnace, 40 feet long and 9 feet wide, is stated to have been built for experimenting with flue dust and slag. The furnace is running successfully, and takes about 15 per cent of its charge in flue dust daily. Slag flows into the furnace from the settler, passes through it, is tapped out at the lower end into cars and sent to the slag dump. The slag tapped out contains about 0.2 per cent less copper than it did when the tapping was done directly from settlers. The flue dust, fed into hopper bins located above the upper end of the reverberatory furnace, is sprayed over the stream of slag by means of an oil-burner blast. It is distributed over the slag in the upper end of the furnace, and runs along with the stream, disappearing at about the center of the furnace. The cost of operation of the furnace, which is, however, built as an experiment, and, therefore, rather crude, is stated to be about $80 per day, including oil, blast and labor. About 350 tons of slag is stated to pass daily through the furnace.

The Washoe Plant of the Anaconda Copper Mining Co. in 1905.—A lengthy paper by L. S. Austin in the *Bi-Monthy Bulletin* of the American Institute of Mining Engineers gives a detailed description of the equipment of the large Washoe plant of the above company in Anaconda, Mont. The company is controlled by the Amalgamated Copper Co. The plant in 1905 handled 7,000 tons of ore daily, with an output of copper of 500,000 pounds. The consumption of coal was 600 tons, of coke 400 tons, and of lime-rock 1,600 tons, while 190 tons of flue dust and 9,000 tons of slag and tailings were produced. The blast furnace plant formerly contained seven furnaces,

each 56 x 180 inches in area at the tuyeres. Three pairs of them, including the 21 feet space between each pair, were united, so that there are now three large furnaces, each 51 feet long, and one furnace 180 inches long. At the slag floor level are seven fore-hearths or settlers, two fore-hearths for each of the large furnaces. On a matte track, at a lower level, stand the matte ladles into which the fore-hearths are tapped. The furnaces have side-boshes of 8 inches, making the width at the throat 6 feet. They have no end boshes. The bottoms of the crucibles slope towards the two tapping holes. The top of the furnace is closed, the gases being carried off by three down-takes to a large dust chamber adjoining the blast-furnace build-ing. The number of tuyeres is 88, set at 14-inch center, 46 of them being arranged at the back and 42 at the front side of the furnace. Usually the four tuyeres nearest the ends of the furnace are plugged, as it is found that by this means secretions are somewhat prevented. The bustle pipe extends completely around the furnace, with two branches from the blast main, so that two valves must be closed in order to take the blast off the furnace. The tuyere pipes are 6 inches in diameter, the nozzle having a 1-inch hole. The fore-hearths are 16 feet in diameter and 5 feet high. The dust chamber is 40 feet wide, 280 feet long, and 20 feet high, and is provided with sheet-steel pyramidal hoppers for the purpose of drawing off the dust into the charge cars. The grade of the matte produced is 40 per cent, and the loss in flue dust amounts to 3 per cent. The large furnaces are stated to have the following advantages: Saving in fuel, saving in jacket water, quick and large discharge of matte and slag, decrease of incrustation, elasticity of operation, large flow through fore-hearths, the possibility of varying the slag in the different parts of the furnace, lesser labor cost. The working plant contains 56 McDougall roasting furnaces, as improved by Evans and Klepetko, arranged in 14 rows of 4 fur-naces each. The reverberatory furnace plant contains 7 re-verberatories, which are the largest in the world. Their dimen-sions are as follows: Length of hearth of No. 1, 115.8 feet; Nos. 2, 4 and 5, 102.5 feet; Nos. 3, 6 and 7, 112.5 feet. All hearths are 19 feet wide, and the fire-boxes are 16 feet x 7 feet, or 112 square feet area. The construction and operation of the furnaces are described in detail. The furnaces can smelt from 250 to 325 tons of ore daily, and burn from 55 to 60 tons of coal per 24 hours, or 21 per cent of the charge. About 10 per cent of this is recovered from the cinders as coke. A typical charge consists of calcines, limestone (put into the charge at the McDougall roasting furnaces) and flue dust, containing SiO_2, 26.1 per cent; FeO, 31.3 per cent; CaO, 2.9 per cent; S, 8.1 per cent, and Cu, 9 per cent. The slag contains SiO_2, 37.8 per cent; FeO, 38.6 per cent; CaO, 4.6 per cent; Al_2O_3, 6 per cent, and Cu, 0.37 per cent. The loss of sulphur by volatilization is 33 per cent. The converter plant contains 10 converter stands for converters of the barrel type, the converters being 8 feet in diameter and 121 feet 6 inches long, of 10 tons capacity. The lining consists of siliceous ore, which contains from 70 to 85 per cent of silica, with values in copper, gold and silver. The operation of the converters is also described at length. The last part of the paper gives a brief description of the arsenic plant, the coke washing plant, the slime ponds and the brick plant.

Pyritic Smelting.—As a contribution to the discussion of a paper before the British Institution of Mining and Metallurgy, by Alabaster and Wintle, which was abstracted in the June issue of ELECTROCHEMICAL AND METALLURGICAL INDUSTRY, G. N. Blakemore gives his experiences in the practice of the pro-cess. He claims that hot blast is not of the least commercial use in smelting copper ores which carry up to 20 per cent of sulphur, or any other class of sulphide copper ores. He has saved 7d. per ton of ore by discontinuing the use of hot blast at the furnaces of the Great Cobar Copper Co. Moreover, the furnaces produced a better grade of matte and ran faster on cold than on hot blast. The ore from the Great Cobar mines contains on an average about 12 per cent of copper. The hot

blast had a temperature of 500° to 600° F. By stopping the use of the hot blast on the furnace at the Nymagee mine and increasing the depth of the furnace from 5 feet to 6 inches of charge to 11 feet 6 inches, he obtained a first matte running from 10 to 15 per cent of copper and equal to that produced with the hot blast, while the furnace smelts 220 tons per week more than when hot blast was used. This ore carries up to 18 per cent of sulphur. He advocates large furnaces, a deep ore column and a large volume of air from a positive source, such as, for instance, blowing engines, as they are used on iron-blast furnaces. He recommends the following type of furnace where large ore reserves are available: Twenty feet long by 4 feet at the tuyeres, 16 to 18 feet depth of ore column, water-jacket from feed floor to crucible, volume of wind per minute 30,000 cubic feet of free air, from a blowing engine capable of delivering this quantity against any pressure up to 10 pounds per square inch. A furnace of this kind, he claims. will smelt daily 400 long tons of ore with not more than 20 per cent of sulphur, while the tonnage ought to run up to 550 per 24 hours with ores carrying 30 to 40 per cent of sulphur. He believes that within a limit of 20 per cent silica it is of no importance what the slag carries in silica, as long as it re-mains fluid enough to run out of the furnace and is low enough in copper. He has produced slags with from 21 to 50 per cent silica, the copper contents of which were practically the same.

GOLD.

Tube Milling Practice.—In a paper read before the Chemical, Metallurgical and Mining Society, of South Africa, published in the April issue of the journal of that society, W. R. Dowling describes in detail the plant of the Robinson Deep Gold Mining Co., Johannesburg, and the method of operating the tube mills. The plant comprises eleven classifiers, each 2 feet x 2 feet, of the ordinary inverted pyramid form with sides sloping at 60°. They receive the tailings from 200 stamps, crushing through a 700-mesh screen, as well as the return pulp from the tube mill. The underflow pulp goes to two unwatering spitzkasten, one of which is located at the head of each of the two tube mills. The latter are 22 feet long, with central feed and discharge, the internal diameter of the shell being 5 feet. They discharge onto shaking amalgamating tables, which receive 190 1½-inch shakes per minute, and have a fall of 10 per cent. One mill is provided with four of these tables, each 12 x 5½ feet, while the other has eight small tables of half the length of the first mentioned ones, but the same width. In the operation of the tube mills, according to the author, a most important point is the separation from the battery pulp of that portion which it is proposed to regrind, and the plant is now being increased by half in order to effect a still more complete separation. He also lays great stress upon the necessity of careful sampling for grading, the samples being taken from the overflow of the classifier for the purpose of controling the working. The tube mills are run with the object of reducing the percentage of material remaining on the 60-mesh in the classifier overflow to a minimum. The dewatering spitzkasten thicken the pulp to, say, 40 per cent of water and 60 per cent of solids, and they also exclude a great deal of slime, which is led around the mill in the overflow, to join the effluent material. Another import-ant point in the running of tube mills is the constant care and attention needed to meet any variation in the volume or nature of the battery pulp, in order to run the mills to their full capacity. It is quite easy to run them below that capacity by feeding thin pulp and not much of it. The mills at the Robin-son plant are lined with silex, and the time of stoppage for relining has gradually been decreased from a week to three days four hours, the latter figure including the shoveling out of pebbles and sand after stopping, the hammering out of the worn linings and the introduction of the pebbles and sand after the mill has been relined. The pebbles used are imported from the Danish and French coasts, and have a diameter of from 2 to 3 inches. It has been found that considerable economy could

be effected by using a certain amount of selected banket ore. The pulp after leaving the mill is evenly distributed over the plates by means of a launder, along the heads of the tables, with holes pierced through the bottom. These holes act like spitzkasten, so that the first shaking table gets the richest and heaviest portion of the pulp.

The Homestake Slime Plant.—An article in the *Mining Reporter*, June 21, by H. J. Baron, contains a brief description of the plant at present under construction for the treatment of gold-carrying slimes of the Homestake Mining Co. The pulp from the 4,000 tons of ore daily treated at this plant is, after amalgamation, separated into 2,400 tons of sand and 1,600 tons of slime. The latter carries average values of 80 cents per ton, and is at present allowed to go to waste. The plant is being equipped with filter presses constructed by Mr. C. W. Merrill, the cyanide manager of the company. The presses are constructed so that they can be readily and rapidly discharged automatically, so as to reduce the labor of discharging to a minimum. The following results are stated to have been obtained on a 10-ton capacity press, of 35 tons weight, and fitted with frames of 4 feet x 6 feet in cross-section, which press has been in operation for upwards of eighteen months on slimes carrying values of from 80 cents to $1.20 per ton. On a 131-charge run of 1,291 tons, the charges being all discharged without opening the press, the average assay value of the slime before treatment was 91 cents per ton. The average value of the slimes after treatment was 10 cents per ton, and the extraction by assays per ton treated was 90 per cent. The recovery in precipitate was 83 per cent, or 83 cents per ton. The treatment is simply lixiviation in the press and no agitation. The amount of solution used per ton of slime was 0.73 ton, the amount of water used in sluicing being 4 tons to 1 ton of slime. The thickness of the slime cakes was 4 inches. Twenty-four presses are being installed at the plant, each of which weighs 65 tons and has a capacity of 26 tons to a charge. The strength of solution used will be 1 pound of cyanide to 1 ton of solution. A recovery of 92 per cent of the assay gold values in the slimes is expected to be made, at a total cost of not exceeding 25 cents per ton of slimes treated. The new plant is so arranged that the flow of material throughout is by gravity. It is run by electric power, the power required for operation at a capacity of 1,600 tons daily being estimated at approximately 1/10 hp. per ton of material treated.

Rand Metallurgical Practice.—E. M. Weston, in a paper in the *Australian Mining Standard*, gives a brief review of the results obtained by the Denny Bros. at the new plants erected on the New Gooch and the Meyer & Charlton mines, Transvaal, and discusses the adaptability of a similar process to Western Australian practice. The slimes plant at the above mines consists of three or four conical steel vats, provided with a slow-agitating device, for the purpose of preventing the settlement of the slimes on the sides to too great an extent. The slime is fed by gravity from one vat to another, and the vats automatically thicken the pulp, so that the bulk of the solution is delivered clear enough to be sent to the zinc boxes, while the thickened slime is sent to the filter presses to have the remaining solution pressed out and then to be washed. Cyanide solution is also used in the battery, and most of the gold is dissolved in the boxes and launders before reaching the plant for sands and slimes. A great saving in water consumption is thus claimed to be effected, as compared with the method of using decantation slimes plants and slimes dams, which is quite a factor at plants where water is expensive. The cyanide consumption is low, being 5 pounds per ton treated. The total amount of solution in circulation for an 80-stamp mill is 2,133 tons, which carries 2 dwt. of gold per ton in solution and 0.038 per cent average cyanide. In March the screen assay at the Meyer & Charlton mill showed 10.871 dwt. The average value of the residues from the sands was 14.4 grains, and from the slimes 6.91 grains. There were 70.7 per cent of sands and 29.05 per cent of slimes. The average assay value of the residues was, therefore, 12.39 grains, which is equivalent to an extraction on screen assay of 95.302 per cent. In discussing the applicability of this method in West Australian plants the author points out that the ore in the Transvaal is comparatively easy to be treated, whereas, in West Australia as well as in Cripple Creek the difficulties are much greater. Agitation of the slimes before filter pressing has nearly always been found necessary, and it has been necessary to pass a great proportion of the slimes solution through presses. A plant, such as that designed by the Messrs. Denny might, in cases where the ore was refractory, not allow of sufficient agitation to promote rapid solution.

Notes on Stamp Mill Practice.—In the course of a paper read before the recent meeting of the Canadian Mining Institute, Mr. C. de Kalb gives some interesting remarks about various details connected with stamp milling practice. He deals with screens, foundations of the battery, the stamp duty, mortar liners, shoes and dies, inside amalgamation, the dressing of the outside plates, the use of salts with coppery ores and outside amalgamation. In regard to stamp duty he is of the opinion that this term is relative, depending not only on the character of the ore but on the rate of discharge which has been found to give the most economical results. The metallurgist who has not advanced to the point of tolerating the low-crushing efficiency per horse-power of the stamp mill, for the sake of the extraction by amalgamation which he can make it yield, is in error at the very foundation of the trade. If a large recovery of the gold values inside of the batteries is not obtained, there is something wrong with the methods, or else the stamp mill should not be used, and the whole process needs study and revision. He emphasizes the fact that it should be borne in mind that it is not necessarily the mill man who can put through the largest tonnage, who is earning the most money. Thus, during a recent experience in California the author found that he could obtain a recovery of 91.5 per cent when crushing 0.121 ton per stamp per hour, while by using the same screen (with 0.028 inch-diameter of opening), but readjusting the mill in regard to drop, discharge, etc., the stamp duty could be run up to 0.188 ton per stamp per hour, but the recovery was reduced to 77 per cent. As far as the author's experience with concrete mortar foundations is concerned, it has led him to regard them with disfavor. He has in each instance found the ratio of broken stems to be as 1 to 3 in favor of built-up wooden blocks. To reduce the wear of concrete blocks he has employed, with marked success, a facing of ¼-inch wrought iron sheet, at least 4 inches wider than the width of the mortar base and 2 inches larger. This effectually stops the wear of the sharp edge of the mortar on the concrete if any nuts loosen and furnishes a suitable surface for a rubber cushion.

NICKEL.

Conversion of Nickel Matte into Nickel by Means of Blast Rich in Oxygen.—In *Metallurgy*, Nos. 10 and 11, R. Hesse gives the result of laboratory experiments made in Prof. Borcher's laboratory at the Technical University of Aachen, to convert nickel matte into nickel by blowing air rich in oxygen into the matte. The contents of oxygen in the air were run up to 63 per cent, in order to get as high a temperature as possible in the converter as well as to make the blast more effective to carry out the reaction intended. Incidentally the action of nickel monoxide on nickel sulphide was studied, by heating a mixture under different conditions in the electric furnace. The author found, with regard to the latter question, that even when the temperature is increased considerably, it is not possible to get metallic nickel by the action of NiO on NiS. The experiments have, however, shown that an action of NiO on NiS takes place, notwithstanding statements at other places to the contrary. It begins only at temperatures exceeding 1,400°, but it was not possible to obtain a quantitative result according to the chemical reaction $NiS + 2NiO = 3Ni + SO_2$. In regard to converting nickel matte by blast containing oxygen it

was established by the experiments that it is not possible to perform that operation, on account of the oxydation of the nickel, which in the presence of silica is slagged off as nickel oxide. Whether nickel is set free by reaction between NiS and 2NiO, or by dissociation, it is not liberated as such as long as there is undecomposed NiS present, as Ni and NiS (or Ni₂S?) alloy in all proportions. The richer the alloy is in Ni, the more free Ni is oxidized besides NiS. NiO and Ni, however, alloy also, and therefore it is out of the question to obtain commercial Ni by blowing.

ZINC.

Electrometallurgy of Zinc.—In a paper read before the Sixth International Congress of Applied Chemistry at Rome, E. Ferraris discusses the possibility of zinc smelting in the electric furnace. He gives a brief outline of the various types of electric furnaces proposed, including his own, which has been experimented with for some time at Monteponi. In ordinary furnaces the zinc is distilled and the cinders removed once a day, whereas, in the electric furnace a slag must be formed which can be run off frequently. For that reason an excess of reducing carbon must be avoided, whch would prevent fusion of the gangue of the ore. The author states that this absence of an excess of reducing coal is the weak point in the electric furnace, as the zinc fume is partially reoxidized by the carbonic acid from the reaction of carbon monoxide on the zinc oxide, and the oxidized parts impede condensation of the zinc in the liquid state. The composition of the ore should be corrected so as to produce a readily fusible slag and which does not dissolve the zinc oxide too rapidly. The best slag for this purpose is stated to be the monosilicate of iron and calcium. In order to assure continuity of the work and to make the composition of the slag uniform, at least 25 per cent of normal slag must be added to the ore. The author gives the following heat calculation for the treatment of calcined or roasted ore, containing 50 per cent of zinc, to which 50 per cent of fluxes and slag is added, presupposing that a continuous resistance current is used, passing, as in the Monteponi furnace, through a bath of slag of regulatable thickness, upon which descends a closely-mixed conical charge. The slag and the furnace must be kept at about 1,200° C., in order to extract the zinc and smelt the gangue and fluxes. The heat needed to treat 1 kilogram of ore is stated to be as follows: Reduction of 500 grams of zinc, 665 Cal.; loss by oxidation of coal, 222 Cal.; heat to bring the charge to about 1,200° C., 450 Cal., a total of 893 Cal., or about 1,000 Cal., reckoning loss by radiation and secondary reactions. One steam horse-power develops in 24 hours an energy corresponding to 15.176 Cal. If the efficiency of the horse-power transformed into current and heat is 80 per cent. and there is one steam horse-power applied to the dynamo shaft, this will suffice to treat 12 kilograms of zinc ore per day, or 3,600 per year of 300 days. The author estimates the cost of the effective steam horse-power from a hydro-electric works at 100 lire per year, and, therefore, the treatment of a ton of calamine would cost 28 lire for motive power. The author calculates the following cost of treatment per ton of calamine with an average plant: Motive power, 28 lire; reduction coal, 128 kgs. at 50 lire per ton, 6 lire; electrodes, refractory material, repairs for furnace, etc., 2 lire; labor, etc., 4 lire; a total of 40 lire. As the cost of treatment in ordinary Silesian and Belgian furnaces varies from 40 to 50 francs, according to the cost of coal, it is greater, or at least equal to that with electric furnaces, on the basis of 100 lire per year and steam horse-power. The author is convinced that experiments made on a large scale would solve the question in details. The efficiency of the electric furnace might be increased by filling it with hot calamine after calcination or roasting.

Electrogalvanizing.—The importance of detecting flaws in boiler tubes at an early stage, that is, before any machining or fitting is done on them, was early recognized by the English admirality. In this work it was found that zinc plating greatly facilitates the process of inspection, showing up defects and flaws which cannot otherwise be detected. An illustrated article in the London *Electrician,* of July 20, describes a large electrogalvanizing plant installed at the Thames Iron Works, Ship Building & Engineering Co. for galvanizing a variety of work, principally tubes to admirality specifications. The Cowper-Coles regenerative process is used, in which the zinc is replenished by means of zinc dust, which is a by-product from zinc-distillation furnaces, instead of by rolled zinc anodes. This is effected by having a filter bed composed of zinc dust and coke arranged in a separate tank, through which the solution is continuously passed by means of a centrifugal pump, the anodes being of lead. When zinc anodes are employed it is found that only a small portion goes into solution, the greater part crumbling away; the adoption of a filter bed prevents the waste that would otherwise take place, as the zinc is caught by the filter and gradually dissolved, and the solution kept free from particles in suspension, which have a very detrimental effect on the deposited zinc.

IRON AND STEEL.

Utilization of the Blast Furnace Gases for Power Purposes.—In our Vol. III., p. 95, F. du P. Thomson, who was connected with the pioneer gas power plant of the Lackawanna Steel Co., gave a review and very conservative estimate of the commercial possibilities of blast-furnace gas for the development of power. In our Vol. III., pp. 150 and 190, A. J. Rossi gave further figures on the same subject, on the basis of the results obtained in practice by various engineers. A further valuable contribution to this subject, and one which has not yet been mentioned in these columns, is a paper read by H. Freyn at the end of last year, and published in full, with the discussion, in the February issue of the *Journal of the Western Society of Engineers.* Since this subject is at present of great interest, in view of the activity of the United States Steel Corporation in this field, the conclusions of Freyn shall here be given. He finds that a power plant of about 10,500 hp. capacity, complete in every detail and installed in connection with a blast-furnace plant, would be capable, when running at full-load capacity, of producing 1 hp. per year at the cost of $17.88, no value being placed on the blast-furnace gas. The enormous saving as compared with the production of power in a steam engine plant is still more striking, when the cost of generation of electric current is considered. According to Freyn's tables, 1 kw-hour at full-load capacity of the plant could be produced at 2.95 mills, which is away below the best figure ever reached with a steam engine power plant. Even under worse conditions, that is, when the power plant is running on an average of only 50 per cent of its total capacity, the cost of generation of 1 kw-hour is but 5.50 mills. (For comparison, F. du P. Thomson's figures may be given. In a plant with four 1,000-kw. units, with 50 per cent overload capacity, a load factor of 50 per cent and 24 hours power, the total cost per kw-hour is 1.475 cent; in a plant with four 1,500-kw units, no overload capacity, with a load factor of 80 per cent, 24 hours power, the total cost per kw-hour is 0.630 cent.) An eventual increase in the capacity of the power plant would still tend to reduce the cost of the generation of power per unit, as certain expenditures for the power plant of 10,500 hp. would remain unchanged for additional power units. Figures are finally given of the actual results obtained in the works of the John Cockerill Co., in Belgium, which operate at present seven blast furnaces of about 1,200 tons daily capacity. To produce the power for the electrical service, they had installed 1,000 kw. in steam engines and no gas engines in 1900, and 800 kw. in steam engines and 2,900 kw. in gas engines in 1905. The total operating cost was 157,463 francs in 1900 and 206,328 francs in 1905; i. e., the increase in operating cost amounted to 31 per cent only, whereas, the capacity of the power plant had been increased 370 per cent. The cost per kw-mour fell from 0.088 franc in 1900 to 0.0206 franc in 1905,

so that the cost of 1 kw-hour in 1905 was but 25 per cent of the corresponding cost in 1900.

Corrosion of Iron Wire.—*Farmers' Bulletin,* 239. of the Department of Agriculture, contains an account of an investigation by A. S. Cushman, on the corrosion of fence wire. It seems certain that modern Bessemer and open-hearth steel wire rusts much more rapidly than the iron wire made twenty or more years ago. The author produces some evidence to show that manganese, especially if it is unevenly distributed in the steel, is at least in part cause of the trouble.

Cleveland Convention of American Foundrymen's Associaton.—A very full and profusely illustrated account of the proceedings of the recent convention of the American Foundrymen's Association may be found in the July issue of *The Foundry,* from which we abstract the following papers which are metallurgical interest:

Fluxes in the Cupola.—N. W. Shed pointed out that the value of fluxes in the cupola is not generally appreciated by foundrymen. There are two available fluxes for the cupola, namely, limestone and fluorspar, but limestone is far cheaper and far better than fluorspar as a flux. The most practical and easily fusible slag is a monosilicate, which means equal amounts of silica and alkaline bases. It is a good rule to figure the limestone on the weight of the coke, using 25 per cent limestone. This amount will flux any ordinary coke ash with the average amount of sand on pig and scrap. If the amount of sand on the pig is excessive, figure 30 per cent limestone on the weight of coke. With a low coke ash, machine pig and clean scrap, the limestone may be reduced to 20 per cent and make a good cinder.

High-Grade Ferro-Silicon and Cast Iron.—A. E. Outerbridge, Jr., discussed the beneficial effects of adding high-grade ferro-silicon to cast iron. He first referred to an important problem which comes up daily in foundry practice, namely, to provide metal melted in one cupola and one heat, suitable for a great variety of castings and requiring wide variations in physical properties and chemical composition. The author's practice is to group all the small work for the day, needing soft ductile iron, so that it will be cast in the beginning of the run of iron from the cupola which is, of course, charged with the required amount of the softest grades of iron; the proportion of silicon in such mixtures approximating 2½ per cent. This is followed by a medium grade of iron, calculated to contain about 1.9 per cent silicon, suitable for miscellaneous castings of medium weight and thickness. Finally, come strong iron mixtures for large and heavy work. They may contain as little as 0.5 per cent silicon, and may possess high chilling property. The author refers to some special castings which have been made in the foundry of Wm. Sellers & Co., and which have united the two antagonistic properties of high tensile strength with softness and ductility in a remarkable degree. Tensile tests of bars 0.5 inch square section formed in the same mold with the castings and attached thereto, have shown, when turned and pulled on the 100,000-pound Emery hydraulic testing machine, extraordinary tensile, viz.: 40,535 to 44,565 pounds per square inch. The element silicon is practically the governor which determines the degree of hardness of cast iron. The author made a long series of experiments, both with commercially pure metallic silicon and with ferro-silicons of various concentrations. Pure metallic silicon was found impractical, partly on account of the high melting point and partly on account of its levity, which causes the powder to float to the surface of the molten iron if sprinkled into the ladle, or to simply cake if fastened to the bottom. Very good results were obtained with a ferro-silicon containing 50 per cent silicon, which in the form of a coarse powder was sprinkled into the molten iron as it was tapped into the ladle. The author speaks well of the chill cup test, which is in effect a physical analysis of practical

value, which enables the foundry foreman to know the character of every ladle of iron, small or large, before he pours his metal. The chill cup tests of the treated and untreated iron showed at once that the small amount of 50 per cent ferro-silicon added (half a pound in a 200-pound ladle) has dissolved in the iron, wiping out the chill in the metal almost completely. The metal of the treated bars was much softer in appearance, as was already indicated by the chill cup tests, but, instead of being weaker, the treated bars were all stronger than the untreated bars. In other words, increased softness and increased strength are here combined. The rationale of the increase in strength with increased softness is probably to be found in the deoxidation of the iron by the silicon added in its nascent condition.

Thermit in the Foundry.—W. M. Carr discussed new applications of the thermit process in foundry practice. In a number of cases where it was important to pour light, thin sections, and where the metal was rather dull, the desired increase in temperature and fluidity was gained by an addition of thermit placed in the bottom of each hand ladle, into which the iron was poured from the bull ladle. In this way the castings were free from cold sheets. In another case a foundry was equipped with a single 2-ton Tropenas converter and only one 5-ton ladle. It was desired to pour some ingots and castings requiring twice the capacity of the converter, so that it was necessary to make two heats separately and pour them both into one ladle. The evident difficulties of this method of making "double blows" were overcome by keeping the first charge of steel hot and liquid by the insertion of thermit cartridges attached to iron bars and plunged to the bottom of the ladle at regular intervals during the progress of the second heat, which occupied about 45 minutes. The result was entirely satisfactory. In the treatment of large headers—both iron and steel—the application of thermit in suitably sized cartridges has kept them open longer and produced a better feeding effect than any other means. In the manufacture of semi-steel castings, two uses of thermit have been developed. First, when the necessary steel is melted in the cupola with regular charge, the insertion of a semi-steel cartridge in the ladle produces a poling action, which thoroughly mixes the metal with no loss in temperature. Second, if it is desired to make but a few semi-steel castings among a larger number of ordinary castings, steel borings or chips can be put in the ladle bottom, the iron tapped in, and by the insertion of a semi-steel cartridge the steel will be melted and thoroughly and completely dissolved in the iron. This obviates the necessity of making special heats for occasional semi-steel castings. The author then spoke briefly on repairs to gray-iron and steel castings.

Electric Smelting of Iron Ore at Sault Ste. Marie.—A paper by Dr. E. Haanel, in the August issue of the *Journal of the American Chemical Society,* contains some details which were not covered in his preliminary report, which was the subject of various articles in this journal. These details relate to the construction of the contact between the electrodes and the steel shoe. and to some observations made during the experiments. Thus some of the runs made showed occasionally a rapid temporary decrease of ohmic resistance shortly after the current was put on, when the charge consisted of conductors in the form of small pieces, which phenomenon has recently been studied by Bronn (see p. 148 of our April issue). This trouble occurred chiefly at the beginning of a new experiment, before the furnace had acquired its normal temperature. By adding a few shovels of iron ore, omitting flux and charcoal, it was sometimes possible to cause the electrode, which to keep the current constant, had, by hand regulation, been elevated to return to its normal position. In a few instances, however, this method failed, especially when the furnace was choked with fines and the gases evolved escaped under great pressure. If the explanation offered by Bronn is correct, that the de-

crease of the ohmic resistance during the "Anheizungsphase" is occasioned by the more intimate contact of the conducting pieces of the charge, due to pressure of escaping gases from the pores of the carbon and the carbon dioxide evolved from the limestone, it is evident that preheating the charge, which might be effected by utilizing the carbon monoxide resulting from reduction, would entirely overcome this difficulty, if the charge were sufficiently porous to permit the gases evolved to escape at low pressure. Under such conditions the electrode would maintain its normal position throughout the operation, requiring to be lowered only to keep step with its consumption. In a general comparison with the blast furnace, Dr. Haanel points out the following advantages of the electric furnace: Original small cost of furnace, absence of bulky or costly charging machinery, small expense involved through break-down, small cost and ease with which repairs may be made, no serious complications arising from scaffolding, loss due to wrong composition of charge reduced to a minimum, perfect control of the temperature in the reducing and melting zone.

BOOK REVIEWS.

HANDBUCH DER ANORGANISCHEN CHEMIE. In four volumes. With the coöperation of others, edited by Prof. R. Abegg. Vol. III., part 1. Leipzig: S. Hirzel. 466 pages. Price (in New York) $5.60.

Among the special features of the program of this new, voluminous German handbook of inorganic chemistry are the endeavor to cover fully the results of physico-chemical research; to state not only facts, but to give at the same time the theoretical connection between chemical phenomena; to exercise critical judgment in giving figures and other data. The editor of the handbook is Dr. R. Abegg, professor of the University of Breslau, and editor of the *Zeitschrift für Elektrochemie*, the organ of the German Bunsen Society.

The chemical elements will be covered in three volumes, divided into eight parts, each of which relates to one group of the periodic system. In addition, a special volume will cover general principles. The programme is, therefore, as follows Vol. I., general principles; Vol. II., part 1 and 2, elements of the first and second group of the periodic system; Vol. III., part 1, 2, 3, elements of the third, fourth and fifth groups; Vol. IV., part 1, 2, 3, elements of the sixth, seventh and eighth groups.

Vol. II., part 2, and Vol. III., part 1, have so far been published. Vol. III., part 1, covers boron, aluminium, the rare earths. etc. For each element a critical discussion of determinations of its atomic weight is first given. This is followed by a concise summary of the preparation and properties of the element and its compounds, with a rather full bibliography on the subject. When completed this new handbook will be a very useful reference work, since the endeavor to give only reliable information is plainly evident.

THERMODYNAMIK TECHNISCHER GASREAKTIONEN. Seven lectures by Prof. F. Haber. Munich and Berlin: R. Oldenbourg. 296 pages. Price (in New York) $3.35.

This book is a treatise of thermodynamics of reactions between gases. "It has been written not for the sake of the theorist, but for the sake of the engineer." Knowledge of chemical technology is presupposed, also knowledge of the first elements of calculus. But the principles of thermodynamics are developed, explained and illustrated, beginning with the very foundation.

The first lecture deals with the "latent heat" of chemical reactions and its relation to the energy of reaction. Prof. Haber adopts Helmholtz' conception that a chemical reaction involves a certain latent heat; the two terms of the "Gibbs-Helmholtz equation" for the total energy are, therefore, not called free and bound energy, but energy of reaction and latent heat. It would appear that in this way Haber succeeds in facilitating the fundamental conceptions.

The second lecture deals with entropy and its rôle in reactions of gases; the third lecture gives another derivation of the fundamental formulas and a discussion of their relation to reactions between solid substances. In the fourth and fifth lectures examples are given of reactions taking place without a change of the number of molecules and with such a change. The sixth lecture deals with the determination of the specific heats of gases, the seventh with the determination of the equilibrium of gases, with a discussion of the theory of various engineering problems.

In the presentation of the principles of thermodynamics, Dr. Haber manifests much individuality in details, but he follows in principle the safe and sane orthodox methods of Clausius, Helmholtz. Van't Hoff and Planck, and his book is free from the heresies of modern "energetics." This is most fortunate. While Planck's work will remain the classical treatise of thermodynamics for physical chemists, Haber's book is worthy a similar place of honor in the library—or rather on the desk—of the chemical engineer. His book is not a dry presentation of principles, since the principles are constantly illustrated in a most interesting manner by applications to engineering problems. It cannot be too often said that the only way of getting a working knowledge of principles is by means of applications—not qualitative but quantitative applications. For the engineer a general knowledge of a principle is good for almost nothing, if it does not enable him to base on it a numerical calculation.

Some of the subjects dealt with in Haber's book are of great actual interest, like his calculations on the fixation of atmospheric nitrogen. Other engineering problems which are fully discussed, are the Deacon chlorine process, the contact process of making sulphuric acid, the manufacture of water gas, etc.

The book is dedicated by the author to his wife, Mrs. Clara Haber, Ph. D., "*zum Dank für stille Mitarbeit.*"

Pebble Mills.

Practice has established the fact that pebble mills are the only apparatus whereby all kinds of drugs, chemicals, colors, enamels, glazes, ores, etc., can be reduced to an absolutely impalpable powder without bolting. The pebble mill pulverizes all of the material charged into it and delivers a uniform product. There are no contaminating surfaces; there is no dust in the work room, while the saving in power, labor, space and time is considerable without any loss of the material.

While other mills either crush, twist or cut the material, pebble mills grind principally by friction, the effect being produced by the sliding, tumbling and rolling inside of the mill of a great number of flint pebbles or porcelain balls, which are mixed with the substance to be ground, and the movement being caused by revolving the mill at a regulated speed. However, there is also the grinding effect due to the direct impact of the pebbles falling down upon the material after they have been carried up by the revolution of the mill. Whether the friction or the impact of the falling pebbles will prevail depends on various circumstances, especially on the speed of revolution, the height to which the mill is filled with the pebbles, and whether dry or wet grinding is employed.

In any special case the best conditions for the working of the pebble mill must be found by experiment.

The Abbé Engineering Co. of New York City, who have done, during the past years, a great deal towards the present perfection of the pebble mill, have for this reason installed a complete testing laboratory where they have upwards of a dozen of their different mills constantly set up ready to cut, grind or mix either dry or wet product. In this testing

laboratory the company is enabled to determine the best conditions of operation.

While the impact of the falling pebbles has some effect, yet in most cases the grinding by friction is prevailing. Since a great number of pebbles or balls are used in a minute, every two of which can be considered an individual small grinding mill, there is an enormous grinding surface within a limited space. The simplicity of the operation of the mill results in the necessity of using only unskilled labor during ordinary
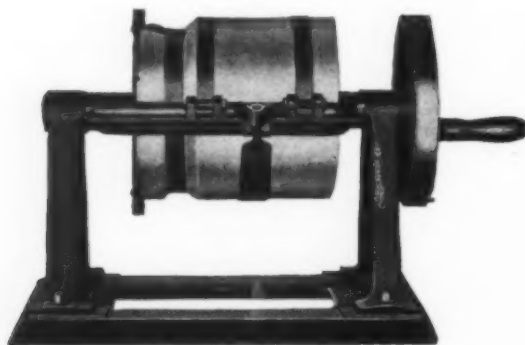

FIG. 1.—"THE LITTLE TROJAN."

operation, attention being needed only while charging and discharging.

No parts require dressing or sharpening, as in all other grinding mills, as practically there is scarcely any wear; consequently the mills are always in good working order.

In the following we describe some types of pebble mills made by the Abbé Engineering Company. All their pebble mills are lined with the finest quality of vitrified porcelain, thus presenting a grinding surface which will neither contaminate nor discolor the material being pulverized. In the smaller sizes the lining is made in one piece, but in the larger pulverizing cylinders the lining is furnished in the form of blocks, which are fitted perfectly into the inner circumference of the cylinders, and cemented in place with the finest quality of cement.

Instead of vitrified porcelain, the Abbé Engineering Co. furnishes, if desired, either natural flint (silex) or wood linings. When the pulverizing cylinders are to be used for the grinding of white enamel, glaze, etc., they furnish a colorless solution to be mixed with the enamel and used in the same way as regular cement when lining the mill.

Porcelain presents the best possible grinding surface, and with ordinary usage and proper care such linings will wear for many years, the exact life of a lining being determined by the variety of the material being pulverized.

These machines will pulverize either wet or dry material, and will pulverize all of same, delivering a perfectly uniform product without bolting. As a mixer, the pebble mill is, without any exception, the best thorough dry or wet mixer in the world, and is, therefore, largely used in the drug business and in the cement manufacturing trade. For mixing colors it is unsurpassed.

Besides the linings there are few things in a pebble mill of greater importance than the quality of the pebbles used in the cylinders. Soft or brittle pebbles not only wear out very rapidly in the machine, but also deteriorate the quality of the material being pulverized, thus making them costly at any price. Pebbles of uniform shape (round or oval) are also far preferable to those of irregular shape, and greatly increase the grinding capacity.

The great simplicity of operation may be seen from the following rules which cover the full directions for operating the Abbé pebble mills.

The dry grinding mills, after having been mounted on brick, stone or iron frame, and encased in such a manner that the bearings are outside of the casing, should be cleaned out thoroughly. This can be accomplished in the best manner by putting in a charge of pebbles (sufficient to half fill the cylinder), and with them a charge of sand (enough to fill all the space between the pebbles). Then close the manhole with the tight cover and revolve the mill at its rated speed. In this way all the cement on top of the lining will be ground off and the pebbles are cleaned at the same time.

The mill should be run for several hours, but as soon as it is clean, which can be determined by looking into it once in a while, can be emptied by replacing the tight cover with the grate discharge cover and the cylinder set running, by which operation the sand is sifted through the openings in the grate and the pebbles or balls are retained. It will take from 2 to 5 minutes to discharge, after which the grate is taken out and the machine is ready for a charge of the material that is to be reduced. Now the mill must be run until the product is of the fineness desired, at which time the same method must be followed to discharge, as stated before, and when the mill is empty it is ready for the second charge.

With wet grinding mills the same directions must be followed, only they require no casing and the operation is the same as above, except that when discharging, a tight cover, provided with a valve, is used instead of the grate discharge cover. The cylinder is turned with the valve downward, and the latter opened so as to allow the material to run into any receptacle placed to receive it.

Another point to which attention may be called is that the pebbles should not be shoveled into the mill, but put in carefully by hand. Figs. 1 and 2 show two types of jar mills of the Abbé Engineering Co. The porcelain jars of these machines are imported. They are manufactured from the finest raw materials obtainable, made in the plastic state, thus forming jars that are impervious to the action of even such materials as ink. These jars are of so superior a quality that constant use of them has never worn them out.

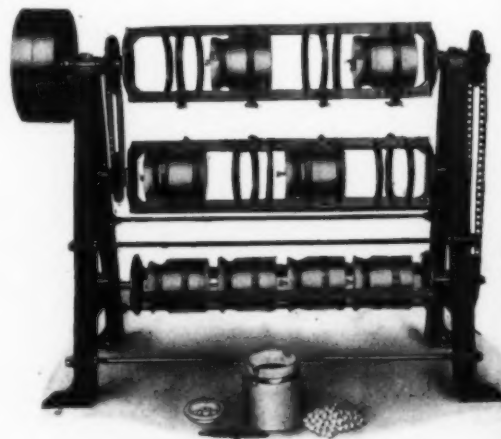The only protection necesary is that in charging the balls

FIG. 2.—TWELVE-JAR LABORATORY MILL.

or pebbles should not be thrown in, since this might result in a break of jars. The balls or pebbles should be carefully put in by hand.

Fig. 1 shows the type called by the company "The Little Trojan," which has a shipping weight of 90 pounds, and requires a floor space of 10½ x 25 inches. The normal speed of the mill is 60 r. p. m. The jars measure outside 8.75 x 9.65 inches (in any case of giving dimensions in this article the diameter is given first and then the length). This jar

is capable of handling from a few ounces up to 5 pounds at a time.

Of course, it is possible to use a battery of such jars and revolve all the jars from the same driving shaft or motor. The advantage of using a battery of several jars is that a different material can be ground or mixed in each jar at the one operation, thus four, six or a dozen products can be handled at a time. The same quantity of material need not be put into every jar, so that one can reduce an ounce of one, a pound of another and 5 pounds of a third.

There is one ceramic color and chemical manufacturing firm in the United States using more than 250 of these mills. This alone will go far towards showing the usefulness of these laboratory mills.

The Abbé Engineering Co. builds a battery of jars up to the number of six in one row, while for a greater number the arrangement is shown in Fig. 2. This shows a twelve-jar laboratory mill, the dimensions of all the jars being outside, 8.75 x 9.65 inches. The floor space required is 6 feet 6 inches by 2 feet 6 inches. The normal speed is 60 r. p. m., and the shipping weight of the whole mill is 1,000 pounds.

Fig. 3 shows a pebble mill, the outside dimensions of the cylinder being 30 x 30 inches. Its normal charge, taking sand as unit, is 200 pounds. Its shipping weight is 2,500 pounds, the floor space required 5½ x 3¾ feet, the normal speed, 38 to 44 r. p. m. With dry grinding this mill requires 1½ hp. With wet grinding ¾ hp.

Fig. 4 shows a pebble mill with the latest improved manhole frame and covers, made by the Abbé Engineering Co., with swing bolts and thumb nuts, as used on all mills designed to handle from 500 to 4,000 pounds of charge when sand is taken as unit. The manhole frames are made with detachable flanges, so that they can be easily replaced with new pieces of flat sheet iron when the inside lining is worn out. These are the flanges that hold the lining in position, and gradually wear down with same. To users of these mills this improvement will readily appeal as an important factor, as it avoids the riveting in of new manhole frames when a machine is to be relined.

The illustration also shows how it is possible to attach a stuffing box, with pipe connection, for steam or air inlet on one

Outside Dimensions of Cylinders	Charge, Taking Sand as Unit	Floor Space Required	Power	
			Dry	Wet
6　x 8　ft.	40 0 lbs.	15　x 10　ft.	18 h. p.	9 h. p.
6　x 5　ft.	2800 lbs.	13½ x 10　ft.	12	6
5　x 4　ft.	1500 lbs.	10½ x 7　ft.	8	4
4½ x 3½ ft.	800 lbs.	9　x 6　ft.	5	2½
3¾ x 3½ ft.	500 lbs.	7½ x 6　ft.	4	2
3　x 3½ ft.	300 lbs.	6¾ x 4½ ft.	2	1
30　x 30 in.	200 lbs.	5¾ x 3¾ ft.	1½	¾
30　x 19 in.	120 lbs.	4¾ x 3¾ ft.	1	½

These pebble mills are undoubtedly destined to play a very important part in chemical and metallurgical arts. In the cement industry as well as in the cyanide process for winning
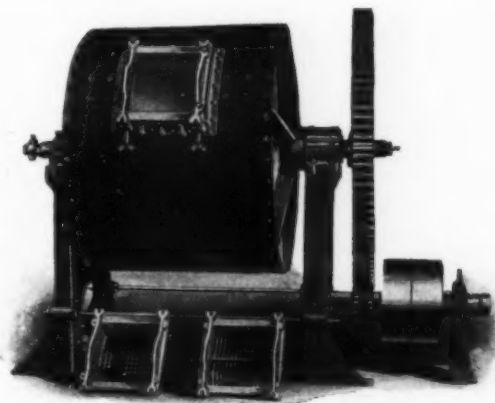

FIG. 4.—LATEST TYPE OF PEBBLE MILL.

gold, the pebble tube mill has found many applications in the past, and is constantly extending its sphere of usefulness. A description of Abbé pebble tube mills paying particular attention to their special spiral feed and discharge arrangement may be found in our Vol. III., page 41.

The Electrochemical Department of the Siemens & Halske Company.

Last year the old well-known Berlin works of the Siemens & Halske Co., in the Markgrafenstrasse, were removed to a magnificent large new plant outside of Berlin, at the Nonnendamm, midway between Spandau and Berlin, West. This new plant is officially named the Wernerwerk, in honor of the late Werner Siemens, and is mainly devoted to what the Germans call Schwachstromtechnik, namely, telephone and telegraph engineering, construction of instruments, etc. The Wernerwerk, however, also contains the electrochemical department. The following description is taken from a recent profusely illustrated book on the Wernerwerk, issued by the Siemens & Halske Co., Engineer Hans Dominik being the author.

For many years the Siemens & Halske Co. have paid special attention to electrochemical developments. They took part in the first development of electrolytic copper refining in Germany, and reference may also be made to the well-known Siemens process for winning copper from ores (our Vol. II., p. 225), which, while not commercially successful itself on a large scale, was instrumental in influencing the later work of Hoepfner, which again started in this country the successful nickel process of the Canadian Copper Co. by its chief chemist, Mr. David H. Browne.

Successful on a very large scale has been the Siemens & Halske cyanide process with electrolytic precipitation in South Africa. This has been the subject of numerous articles in these columns.
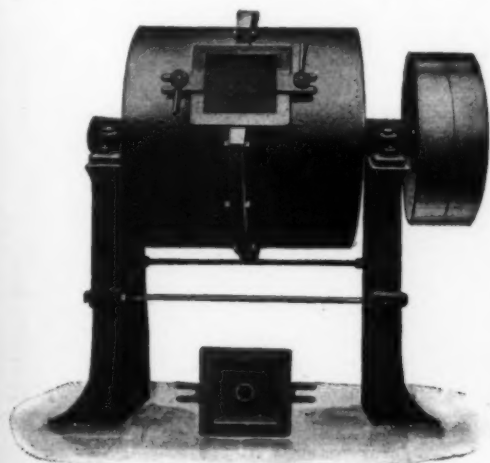

FIG. 3.—PEBBLE MILL.

shaft and a way-cock on the other shaft, to enable grinding or mixing material under pressure.

From the preceding description it will be evident that the simplicity in construction will result in a considerable saving of power, labor, space and time. Concerning power and space required by the Abbé pebble mills the following table will give interesting notes:

ELECTROLYTIC BLEACHING.

The Siemens & Halske Co. has done considerable work in connection with the late Dr. K. Kellner on bleaching with electrolytic hypochlorite. They developed successfully three arrangements, all of which are based on a connection of a larger number of electrolytic cells in series in one apparatus, so as to get the required small voltage at the terminals of each cell from a given electrical supply network at a higher voltage.

In all these arrangements the series connection is obtained by means of bipolar electrodes.

The first system was the so-called point electrode. A thin platinum wire was stitched through a rubber plate, which was then vulcanized, and the wire was cut through at the different stitches and the ends of the wire bent straight. In this way the rubber plate was pierced through by a great number of short pieces of platinum wire ending on both sides of the plate in a point, one side acting as anode the other as cathode. However, the rubber plates were not very durable, and were especially affected by the product of electrolysis on the positive pole.

FIG. 1.—ELECTROLYTIC BLEACHING.

A modification of this method was therefore devised, which consisted in glass plates around which a helix of platinum-iridium wire was wound. These plates were placed tight into recesses of the walls of the electrolytic cell, so that the wire on one side acted again as anode and the wire on the other side as cathode (the American patent of Kellner for this construction was abstracted in our Vol. III., p. 395). This arrangement was, until recently, the normal construction, and has been used in many fiber paper and textile plants, the aggregate electric power used in cells of this type being 2,000 hp. A power consumption of 180 hp. is sufficient for bleaching 10,000 kg. of sulphite cellulose per day.

The third system is characterized by the use of horizontal electrodes, and represents a further increase of efficiency. The arrangement is shown in Fig. 1, the electrolyte passing through the cell along a narrow zigzag path, as shown by the arrows in the illustration. The walls between the zigzag paths divide the horizontal platinum-iridium wire-net electrodes into two halves of opposite polarity. This last system has already found many applications in the textile industry.

In connection with this subject the electrolytic production of hydrogen and oxygen from water may be mentioned, since here also bipolar electrodes are used, to get the required low voltage of 2.5 at the terminals of each cell from the higher voltage of the current supply. The Siemens & Halske Co. uses the apparatus of Dr. O. Schmidt, in Zurich, the mechanical form being similar to a filter press.

The products of water electrolysis, oxygen and hydrogen gases are used for lighting, for filling balloons and for autogenous welding with the oxyhydrogen flame.

ELECTRIC FURNACES.

The Siemens & Halske Co. has bought the Kjellin patents for the electric induction furnace for making steel. Since this process has been described in detail repeatedly in our columns this notice must here suffice.

The Siemens & Halske Co. also has developed several types of calcium carbide furnaces, but the calcium carbide industry has not attained in Europe the importance which had been hoped for. Much is expected, however, from the use of calcium carbide as a starting material for making calcium cyanamide, which can be used directly as fertilizer or may be worked up into cyanide.

For this purpose calcium carbide is used in powder form, and is heated in hermetically closed red-hot iron retorts in an atmosphere of nitrogen. (See the paper by Frank on another page of this issue.)

DISCHARGES THROUGH AIR.

While the production of calcium cyanamide represents one method of binding the atmospheric nitrogen in useful form, the company has also carried out extended experiments on the direct oxidation of the atmospheric nitrogen by means of arc discharges. These experiments have not yet reached finality, but the results are stated to be promising.

On the other hand, the production of ozone by the silent discharge through air has already reached industrial importance. If ozone can be made cheap enough it may be used for many purposes.

At present the strongly oxidizing and sterilizing properties of ozone have been used in practice especially for two purposes. The treatment of ordinary starch with ozone yields a peculiar product, the "ozone starch," which, with its properties, stands midway between starch and dextrine. Like true starch, ozone starch is insoluble in cold water, but dissolves in hot water without forming a paste like dextrine. For this property ozone starch is used extensively as a binding material for paints and colors and for printing on cloth, linen and all kinds of fabrics. For the production of ozone starch there are now in operation two large plants in Kyritz and Fuerstenwalde, in the Province of Brandenburg, in Prussia.

The second important application of ozone is for the sterilization of drinking water, which has often been referred to in these columns. Two plants erected by Siemens & Halske in connection with large municipal water works, were described in our Vol. I., page 179; these are the plants of the cities of Wiesbaden and Paderborn. (It was recently reported that the ozone plant of Wiesbaden had been abandoned, but from a note of Mr. H. P. M. Halbertsma in *Engineering News* of July 12, this is not the case, but the plant is operated each month during one day only, so as to keep it in running order and to preserve it as a reserve supply in case of emergency. For general use the ozonizing process is said to be too expensive, and
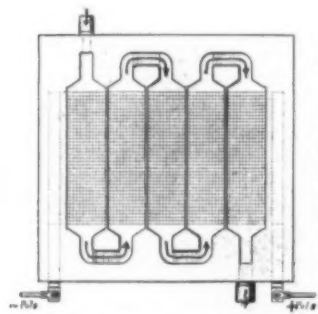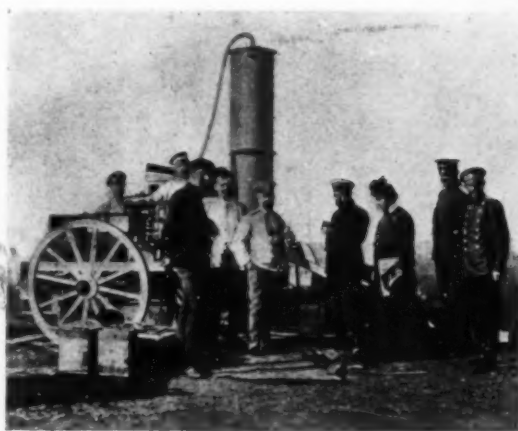

FIG. 2.—PORTABLE OZONE PLANT IN RUSSIAN WAR.

is made so principally by the many bacteriological tests which are necessary for the control of the proper working of the plant.)

The municipal ozone plant of Paderborn was designed for sterilizing 60 to 90 cubic meters of water per hour, and has furnished in continuous operation during day and night the whole drinking water for the city for a number of years. The result has been that the city of Paderborn, which formerly had an epidemic of typhoid fever almost every year, is absolutely free of it now since the ozone plant was started.

Ozone apparatus in smaller size for sterilizing of water in hospitals or in private residences are in use in various cities. Apparatus of this kind have been installed. for instance, in all police stations in Petersburg in the autumn of 1905, when there was the danger of an epidemic of cholera morbus, in order to be able to provide the poor people in case of emergency with pure water.

Fig. 2 shows a transportable ozonizing plant in the form used in the Russian war in order to have always good water for the soldiers.

Portable Voltmeters and Ammeters.

In addition to the line of direct-current switchboard instruments made by the American Instrument Co., attention may be called here to a series of accurate portable direct-current voltmeters and ammeters.

These instruments are built on the permanent-magnet moving-coil type, although very different in important respects from instruments of the same class made heretofore.

By the use of cylindrical steel pivots journaled in the best grade of watch jewels, friction errors. which so often occur because of damage in shipment, or from jars during continued service, are entirely overcome. This method of pivot control is far more difficult to manufacture than the usual conical pivots working in conical jewels; but when once accomplished instruments of this kind will remain practically frictionless for many years.

The permanent magnets in these instruments are made of the best magnet steel obtainable, hardened and aged in accordance with the most improved methods. The moving systems combine light weight with extreme stiffness, and with the method of pivoting already noted it is impossible for the moving coil to come in contact with the pole pieces, or with the core of the magnetic system.

By the selection of a proper winding for the moving coils, "American" instruments have a somewhat larger torque than usual, which permits the use of very strong controlling springs. Because of this fact, and also on account of the unique manner in which current is taken into and out of the moving coils, "zero errors" are completely avoided in ammeters and milli-voltmeters as well as in voltmeters.

PORTABLE INSTRUMENT.

The series resistance in American portable voltmeters is wound so as to be perfectly non-inductive, and each instrument has a negligible temperature coefficient. All voltmeters are adjusted to a uniform resistance of 100 ohms per volt, and, therefore, when multipliers are used they can be interchanged. Also the measurement of insulation resistance and the location of faults and grounds are much facilitated as the necessary calculations are greatly reduced.

The portable voltmeters are self-contained up to and including 750 volts, but any range beyond that can be provided for by the use of external multipliers.

Ammeters are self-contained up to and including 200 amps. When it is desired to use one ammeter with several ranges, the plan adopted is to employ a milli-voltmeter with separate interchangeable shunts. Each milli-voltmeter with its leads has

a resistance of exactly 1 ohm, and will give full-scale defection with a potential difference of 50 milli-volts. All ammeter shunts are adjusted for a potential drop of 50 milli-volts at full load. Consequently, any instrument can be used with any shunt or any number of shunts with one instrument. This is a new feature that will be highly appreciated by engineers who wish to make current measurements through a wide range of values.

"American" portable instruments are made in two types—designated Nos. 4 and 5. Type 4 instruments are provided with special iron shields inside of the wood cases, and can be used close to large generators or under other conditions where an unshielded instrument would be useless. Type 5 instruments lack the iron shield, but in consequence are lighter in weight and somewhat smaller in size.

The new factory of the American Instrument Co. is located in Newark, N. J., but the general sales office is at No. 1114 Chestnut Street, Philadelphia, where Mr. James G. Biddle, President, as well as General Sales Agent of the company, makes his headquarters.

Notes.

ELECTRICAL BOOKS.—The Pratt Institute, of Brooklyn, has issued a list of books on electricity, a special feature being concise annotations concerning the contents and character of the books mentioned.

HYDRO-ELECTRIC INSTALLATIONS.—At the Reliance Works of the Allis-Chalmers Co.. of Milwaukee, there are at present under construction and in various stages nearing completion, forty-three hydraulic turbine units, aggregating 166,000 hp., for shipment to various portions of the United States, Canada and Mexico.

LABORATORY APPARATUS.—The Arthur H. Thomas Co., of Philadelphia, has just issued a second edition of their Catalogue F on laboratory apparatus, especially selcted for laboratories of chemistry and the biological sciences. The catalogue —which is a book of 444 pages—is alphabetically arranged according to subject matters and profusely illustrated. A very full index at the end of the book greatly enhances its usefulness.

ENGINEERS' SOCIETY OF WESTERN PENNSYLVANIA.—The membership list of the Engineers' Society of Western Pennsylvania, corrected to June 30, 1906, has just been issued. The thoroughness with which it has been prepared makes it a model of its kind. The Society has now 892 members; the large majority is located in or around Pittsburg, but the Society has members in other States all over this country and even abroad. This gradual abandonment of the local character of the Society is undoubtedly the reason why, while the absolute number of members attending the monthly meetings has remained about the same, yet when expressed in per cents of total memberships, there is a decrease from 27.8 per cent in 1880 to 4.7 per cent in 1905. The present president is Mr. Julian Kennedy, of Pittsburg.

POROUS DIAPHRAGMS AND CELLS.—The Deutsche Steinzeug-waarenfabrik für Canalisation und Chemische Industrie, of Friedrichsfeld, in Baden, manufacture for electrochemists' use a refractory and acid proof ceramic product known as "Dr. Buchner's Patent Material." Besides the adaptability of this material for evaporating dishes, stills, cooling worms, refractory tubes, etc., it is also made in the form of porous diaphragms. The manufacturers claim that these diaphragms, although somewhat thicker than the ordinary ones, offer exceedingly small resistance to the passage of electric current, and are not easily penetrated by liquids. Furthermore, that their thickness is advantageous from the fact that they thereby resist mechanical influences much better. The resistance of every diaphragm in ordinary electrolytes is determined before

leaving the works, and the determination is guaranteed. They are constructed in any shape desired; e. g., in the form of cylindrical cells, square cells or plates. It is stated that this product is remarkable for its excellent thermo-conductivity, its minute contraction and expansion under the influence of changes of temperature, and lastly, for the fact that it may be made either porous or dense according to requirements. The electric resistance of the diaphragms is claimed to be smaller than that of Pukall and Mettlach cells of the same dimensions. The manufacturers are represented in this country by Messrs. F. Bertuch & Co., Temple Court Building, New York City.

AUTOGENOUS WELDING OF ALUMINIUM.—While the difficulties of soldering aluminium have been overcome by the invention of suitable solders, containing foreign metals (zinc, tin, etc.), yet it has been found impossible in the past to devise a method of autogenous welding of aluminium, by which the two pieces could be so firmly united as to form one solitary unit. Autogenous welding of lead with the oxy-hydrogen flame has long been used to advantage in practice. But all endeavors to develop an analogous process for aluminium failed on account of the well-known property of aluminium to form a fine oxide film on its surface in contact with air; this film may be invisible, but prevents the welding together of aluminium. Mr. M. U. Schoop, in Paris (who contributed an article on autogenous lead welding to our Vol. III., p. 260), endeavored to solve this difficulty by finding a substance which, on the one hand, would protect the heated aluminium surfaces from the air, and which, on the other hand, would dissolve immediately any oxide film, so that an intimate and direct contact of the two surfaces to be welded together is rendered possible. These experiments of Mr. Schoop were successful, so that it is no longer difficult to weld autogenously in a perfect way aluminium of any form, whether sheets, rods, wire, etc., just as it is done with lead with the air-hydrogen or the oxy-hydrogen flame. A burner of special construction is used in which an intimate mixture of the two gases is accomplished before they leave the burner tip. During May last official tensile tests were made by the Laboratoire du Conservatoire des Arts et Métiers, in Paris, which showed that the mechanical properties and resistance of the joints are exactly equal to those of pure aluminium. A specially simple and effective application of this method of welding is the manufacture of aluminium tubes and pipes in any thickness and size, and a company has already been formed in Paris for the production of such tubes, for which there is a large demand in connection with automobiles, balloons, autocanoes and on board ships in general. It is to be expected that the aluminium industry, the growth of which in recent years has been so phenomenal, will thereby find further fields of usefulness.

PERKIN JUBILEE OF THE COAL-TAR INDUSTRY.—We have already alluded in these columns to the fiftieth anniversary, to be celebrated this year, of the epoch-making discovery, by Dr. William Henry Perkin, of the dyestuff "Mauve," by which the foundation was laid for the coal-tar industry. The American "Committee of Fifteen," with Dr. C. F. Chandler as chairman, Mr. A. Kuttroff as treasurer, and Dr. H. Schweitzer as secretary, has arranged a banquet and symposium on the coal-tar industry for October 6. A personal token will be presented to Dr. Perkin, who will attend the celebration. The program also includes the foundation of a Perkin medal to be awarded annually to an American chemist for distinguished work in applied chemistry, and the establishment of a nucleus of a fund for the foundation, at the Chemists' Club in New York City, of a reference and circulating library, covering the entire field of theoretical and applied chemistry, which is to be in charge of a salaried librarian, and to contain duplicate sets, one of them to be used for circulation among American chemists. It is estimated that $5,000 will be amply sufficient to cover the expenses of the personal token and Perkin medal, while the sum of at least $50,000 will be necessary to place the library on a permanent basis.

Personal.

Mr. MAX MEYER, who was connected with Messrs. Eimer & Amend for a long series of years, has recently entered the firm of Ernst Leitz, manufacturers of microscopes and optical instruments, in New York City.

Mr. O. F. SAUGSTAD, formerly connected with the H. & M. Automatic Regulator Co., as master mechanic and superintendent, and also with the Davis & Roesch Temperature Controlling Co., has established offices at 21 Park Row, New York City. He will practice as consulting and advising engineer, making a specialty of temperature and pressure regulation in all their branches.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

OZONE.

(Continued from page 290.)

No. 527,326, Oct. 9, 1894, John T. Donovan and Henry L. Gardner, Springfield, Mass.

Produces ozone, "as much as thirty-eight and five-tenths per cent" at the positive electrode of an electrolytic cell, having separate but communicating chambers containing the electrodes. The electrolyte is a solution of a permanganate of any solid metallic base, such as sodium, silver or barium, but is preferably a saturated solution of potassium permanganate containing crystals of permanganate and 5 to 10 per cent of sulfuric acid.

No. 563,288, July 7, 1896, Walter Lobach, Chicago, Ill.

Oxidizes substances such as oils which it is desired to bleach, by subjecting them to the action of nascent oxygen. The apparatus may consist of spaced electrodes with an intermediate dielectric plate. A high-frequency current is passed transversely between the electrodes, and oxygen or air and the oil to be treated longitudinally between them. In another apparatus, the ozonizer is vertical and comprises a cylindrical electrode surrounded by a dielectric layer, concentrically arranged within a tubular electrode spaced therefrom. The liquid flows down between the two electrodes, which are placed in a chamber communicating with a gas reservoir. Another apparatus comprises a horizontal revoluble chamber having a perforated hollow shaft communicating with the gas reservoir. The liquid to be treated is contained in the cylinder. A series of concentric electrodes, one a rod, the other a perforated tube, extend between the ends of the cylinder and are electrically connected to rings upon which bear brush terminals.

No. 565,952, Aug. 18, 1896, Emile Andreoli, London, England.

A vacuum tube, or one containing rarified gases is provided internally with an electrode consisting of a carbon filament, a wire or several wires, or a bundle of small serrated wire, the points of which face the glass. Outside of, but spaced from the tube, is arranged, longitudinal or spirally, another serrated wire electrode.

No. 568,177, Sept. 22, 1896, Nikola Tesla, New York.

An electric motor of high self-induction is placed in the primary of an induction coil. One terminal of a lighting circuit is connected to one binding post of the motor, and thereby through one field coil, the brushes and commutator and the other field coil, to the brush of an interrupter consisting of a rotating disc with insulating sections. The other terminal of the circuit is connected to another brush bearing on this controller. Around the controller is a circuit of low self-induction, which includes a condenser and the primary of the transformer. The terminals of the secondary are connected to spaced electrodes between which air to be ozonized is driven by a fan.